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# TASK 4 - GEOTECHNICAL DATA BASE
# FOR GROUND CONTROL

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by

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## EXECUTIVE SUMMARY

The stability of an underground opening can have a major impact on a mining operation. As a first step toward predicting stability, it is necessary to systematically describe the rock mass in which mining will take place. This involves three phases, namely; 1) the identification of significant properties; 2) methods of measuring these properties; and 3) quantification of the properties in a manner suitable for input to design.

Experience has shown that the rock properties can be divided into two groups. The first group contains those properties that describe geological aspects of the orebody and the surrounding country rock, in particular the geometry of the different rock units and planes of weakness inherent within them. While the second group contains those properties that describe mechanical aspects of the orebody surrounding rock.

A complication exists due to the effects of mining. Strains are induced in the rock mass as the ground responds to the imposed loadings. These strains can result in the extension of existing discontinuities or the generation of new ones. Thus, the properties of the rock mass may be irreversibly altered as mining progresses. It is very important that such changes to incorporated within any program of rock mass characterization.

Studies at the Thompson Mine have indicated that the single most important geological parameter, apart from rock type, is discontinuity spacing; particularly the spacing of the discontinuity set (foliation) that parallels the orebody. Observations have shown that where spacing is small there is an increased incidence of ground control problems. Data collection procedures have therefore been directed toward obtaining information that will allow spacing to be adequately characterized. Initially, standard data collection methods developed for open pit studies were used. These were found to be untenable in the underground environment, largely due to time requirements. Ultimately, it was found that a simplified version of scan line surveys provided the most appropriate method of data collection.

Because spacing is a three dimensional property its value is dependent on the direction of the line in space along which measurements are made. Though corrections are made to obtain true spacing the use of block volume proved to be a useful unbiased descriptor. Block volume is derived from the matrix block bounded by discontinuities from the three most frequently discontinuity sets. By taking due cognizance of the inherent natural variability of discontinuity spacing, the distribution of block volume can be estimated. This distribution is akin to the particle size curves that form the basic classification system in soil mechanics. By also calculating the shape factor for these matrix blocks a relatively unique characterization of the discontinuities within a rock mass can be obtained.

It is believed that this approach offers a number of advantages over existing methods and will provide a useful addendum to existing rock mass classification techniques. Ultimately the determination of block volume and shape could be used to construct a model of mine openings as an aid to stability evaluations. Further research is required to develop this approach, particularly with the view to determining relationships between block size and shape, and types of failure. If such a relationship could be established it would greatly assist in the prediction of stope and drift stability.

## SOMMAIRE EXÉCUTIF

L'état de stabilité des chantiers ouverts souterraines peut avoir un impact majeur sur les opérations minières. Comme première étape pour prévoir la stabilité des parois rocheuses, il est nécessaire de décrire systématiquement le massif rocheux où l'excavation aura lieu. Ceci implique trois phases: 1) l'identification des propriétés significatives; 2) des méthodes pour mesurer ces propriétés; et 3) quantifier les propriétés de manière à ce qu'elles soient utilisables pour le design des excavations.

L'expérience démontre que les propriétés du massif rocheux peuvent être divisées en deux groupes. Le premier groupe contient les propriétés qui décrivent l'aspect géologique du gisement et de la masse rocheuse avoisinante, en particulier la géométrie des types de roches et des plans de faiblesses propres entre eux. Le second groupe inclut les propriétés qui décrivent l'aspect mécanique du gisement et de la masse rocheuse qui le contient.

Certaines complications se présentent lorsque le minage de ces zones s'effectue. Des déformations sont induites dans le massif rocheux car le terrain répond à la surcharge imposée. Ces efforts peuvent résulter dans l'extension des discontinuités existantes ou à la génération de nouvelles fissures. Ainsi, les propriétés de la masse rocheuse peuvent se détériorer de façon permanente lorsque le minage progresse. Dans ce cas, il est très important que cette détérioration des propriétés mécaniques des roches soient incorporées dans tout programme de caractérisation des massifs rocheux.

Des études effectués à Thompson Mine ont indiqué que le paramètre géologique majeur, en excluant le type de roche, est l'espacement entre le réseau de discontinuités; particulièrement les discontinuités (foliation) qui sont parallèles au gisement. Des observations ont démontré que lorsque l'espacement est faible, il y a un accroissement des problèmes de contrôle de terrain. La procédure de récolte des données a donc été orientée de façon à obtenir une information adéquate sur les caractéristiques des espacements. Initialement, la méthode de récolte de données développée pour les études effectuées dans la fosse de Thompson Mine fut utilisée en souterrain. Mais, cette méthode a vite été abandonnée car elle exigeait un temps considérable pour être exécutée. Finalement, une version

simplifiée de l'arpentage des lignes fut plus appropriée pour la récolte des données.

Vu que l'espacement entre les discontinuités est une propriété tri-dimensionnelle, les valeurs sont dépendantes de la direction des lignes le long desquelles les mesures ont été prises. Bien que des corrections soient apportées pour obtenir l'espacement réel entre les discontinuités, l'utilisation du volume des blocs a démontré être un descripteur non-biaisé. Le volume des blocs provient des trois réseaux de discontinuités les plus fréquents qui définissent les frontières du bloc. En prenant pour acquis que l'espacement entre le réseau de discontinuités est une variable aléatoire de type normale, la distribution de la grosseur des blocs peut être estimée. Cette distribution est similaire à la courbe granulométrique utilisée dans la classification des sols en mécanique des matériaux meubles. Une caractérisation unique des discontinuités de la masse rocheuse peut être aussi obtenue en calculant un facteur qui décrit la forme du bloc de la roche.

Cette approche offre de nombreux avantages sur les méthodes déjà existantes et elle fournira un ajout utile aux techniques déjà existantes de classification des massifs rocheux. Finalement, la détermination du volume et de la forme du bloc pourraient être utilisés pour produire un modèle d'exploitation à chambre ouverte pour fournir un appui à l'évaluation de la stabilité des excavations. Une recherche supplémentaire serait requise pour développer cette approche, particulièrement pour déterminer la relation entre la grosseur et la forme des blocs et le type de rupture. Si cette relation pourrait être établie, elle aiderait grandement la prédiction de la stabilité des chambres ouvertes et des galeries.

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## 1.0 INTRODUCTION

The stability of an underground opening can have a major impact on a mining operation. As a first step toward predicting stability, it is necessary to systematically describe the rock mass in which mining will take place. This involves three steps, namely; 1) the identification of significant properties; 2) methods of measuring these properties; and 3) quantification of the properties in a manner suitable for input to design.

Experience has shown that rock properties can be divided into two groups. The first group contains those properties that describe geological aspects of the orebody and the surrounding country rock, in particular the geometry of the different rock units and planes of weakness inherent within them. While the second group contains those properties that describe mechanical aspects of the orebody and the surrounding rock. A further division of rock properties can be made within each of the two groups; namely those that influence regional behaviour and those that influence local behaviour. This distinction is important; it determines, by and large, whether the rock mass can be treated as continuum, with behaviour being governed by the global properties of the rock mass; or whether the rock should be regarded as a discontinuum, with behaviour being controlled by the properties of discrete blocks of ground.

A further complication exists due to the effects of mining. Strains are induced in the rock mass as the ground responds to the imposed loadings. These strains can result in the extension of existing discontinuities or the generation of new ones. Thus, the properties of the rock mass may be irreversibly altered as mining progresses. It is very important that such changes be incorporated within any program of rock mass characterization.

A number of procedures for the description of rock masses are available in the literature. These obviously provide a cornerstone on which to develop site specific data collection methods. By and large, however, these procedures were developed for the design of civil structures and are either of a semi-quantitative nature or are extremely time consuming.

Because of the volume of ground excavated in a mine and changes in the rock mass that occurs as a result of mining, data collection must be simple, rapid, and as quantative as possible.

Two approaches to data collection have been examined. The first approach is based on detailed measurements underground of key parameters in order to derive design values. Unfortunately such an approach suffers from the amount of effort required. The second approach is the use of rock mass classification schemes to rate the rock mass according to its "quality". Such schemes are based on measurements of five or six rock mass parameters including number of discontinuity sets, discontinuity spacing, discontinuity condition, intact rock strength, and pore water pressure. The resulting quality rating can then be used to scale design parameters such as rock mass strength or rock mass deformation or can be directly correlated with experience.

Building on the two approaches, the concept of a "matrix block" was evolved. The matrix block describes the size, shape and orientation of blocks of intact ground, bounded by the jointing, that is pervasive within the rock mass.

## 2.0 OBJECTIVES

### 2.1 Ground Control Philosophy

For any program of rock mechanics to be successful it must predict events. The prediction process is best explained by the simple flow chart, shown in Table 2.1. Three prediction types are defined. These are:

- Type A - Before Event
- Type B - During Event
- Type C - After Event

In the mining environment, Type A events are usually made prior to stoping when no site specific experience is available. Type B predictions are done during mine life and are constantly calibrated against experience. Type C predictions are made after the event (e.g., mine closure) and can be considered as autopsies.

In order to progress from one level of prediction to the next, both the amount and the quality of base data has to be expanded and improved upon. For example, prior to startup of a mine, usually the only available information on rock mass characteristics is from exploration drill core. Such data will generally allow mine wide trends to be identified, and gross rock properties to be defined. However, as the mine develops access will be gained to the various rock units, allowing in-situ inspection and measurements of rock properties. In addition, the response of the rock mass to mining can be monitored, giving rise to site specific experience.

With increasing mining, the importance of individual rock properties to that particular environment is realized, data collection procedures refined (and often simplified), and experience quantified.

Regardless of the particular rock property on which information is being sought, there are two basic levels of data gathering, depending upon the amount of detail required. In a subjective (biased) survey only the observed (as opposed to measured) values of the key parameters are recorded. Judgement therefore plays an important role. In an objective (random) survey, properties are measured along or within a specific sampling line, area or volume.

### 2.2 Statement of Problem

A rock mass is never a continuous, homogeneous, isotropic material, but is intersected by a great variety of discontinuities such as faults, joints, bedding, foliation or cleavage. In addition, there can be a number of rock types, which may have been subjected to varying degrees of alteration or weathering. It is clear that the behaviour of such a material, subjected to external loading, cannot be adequately analyzed or predicted unless these structural features are taken into account.

When subjected to loading, the rock mass behaviour will be strongly influenced by the presence of the weak discontinuity surfaces, with failure tending to preferentially occur along surfaces associated with these discontinuities. In order to establish the mechanism by which movement may occur, and also the resistance to movement, a geotechnical appraisal of the rock mass is required.

Once the spatial relationship of the various rock types and discontinuities have been approximately determined together with their relevant mechanical properties, a "model" can be formed describing the spatial variation in geotechnical properties. Incorporation of mine geometry and stress distribution into this model will enable predictions to be made of the performance of underground openings.

An underground mine consists of a complex geometry of haulages, drifts and stopes. Each is mined at a different period of time creating a constantly changing induced stress environment. This changing environment results in changing rock mass behaviour which is particularly evident as the induced stresses approach, and exceed peak strength. In these instances, new fractures are formed and existing fractures extended, thus irrevocably altering the rock mass characteristics. Therefore, it is not sufficient to only record the in-situ geological properties prior to mining but it is also necessary to continuously monitor changing conditions during mining.

A mining data base will therefore consist of information describing rock mass characteristics and observed conditions underground at various stages of mining. For an average mining operation such a data base will ultimately contain an appreciable volume of information. For the information collected to be useable it must be readily assimilated into a summary description of the distribution of rock quality both on a mine wide basis for long term planning purposes and on a stope basis for specific support design and predictions of stoping efficiencies.

## 3.0 ROCK MASS CHARACTERIZATION

### 3.1 Geological Aspects

The scale of geological features is vast, ranging from the microscopic grain boundaries, for example, to macroscopic features such as faults that can extend over hundreds of miles. This wide range, however, can be substantially reduced when considering the influence of geology on mine stability. Two broad classes of geological features can be defined:

- **Structure**
- **Fabric**

Structure refers to large scale continuous features, such as geological contacts, faults or shear zones, which can be uniquely located in three dimensional space. Thus, structure will tend to naturally divide the rock mass into different zones or regions in which a degree of homogeneity could exist.

Fabric, on the other hand, describes the relatively small scale features that are pervasive throughout the rock mass. Due to the high frequencies of occurrence of elements of fabric it is usually necessary to consider fabric in a statistical manner. Jointing, bedding, cleavage, foliation etc. all make up rock fabric.

Therefore, the basic parameters to be considered when collecting geological data related to mine stability are:

**A. Structure**

- Distribution of major rock types
- Location and orientation of major faults or shear zones, etc.

**B. Fabric**

- Orientation of discontinuities and division into "sets" having similar orientation
- Persistence and spacing of discontinuities within sets.

## 3.2 Structure

### 3.2.1 Distribution of Major Rock Types

Rock type forms the most basic classification process in any rock engineering study. For example, knowledge of whether a hanging wall consists of a schist or a quartzite will immediately give an insight of possible behaviour or failure mechanisms. However, it is not sufficient just to detail lithology. Information must also be obtained on attitude and thickness of individual rock units. Though such data is normally recorded as a matter of course by mine geology departments it is often only available on level plans or cross-sections. It can be extremely useful to present such data in the form of isopachs showing the thickness of the ore, immediate hanging wall rocks and the immediate footwall rocks. The orientation of the reference plane on which the isopachs are constructed depends on orebody geometry; though for the majority of Canadian mines the vertical mine longitudinal will be the most appropriate projection. Further detail on rock isopachs is given in section 6.

### 3.2.2 Location of Major Structures

The presence of thoroughgoing structures can have a major impact on excavation stability. Similar to rock type, such structures are normally mapped and recorded as a matter of course by a mine geology department. Nevertheless it cannot be over emphasized the importance of mapping the location, attitude and size of such features.

## 3.3 Fabric

### 3.3.1 The Concept of a Matrix Block

Rock masses are composed of rock materials and natural discontinuities. The nature, orientation and intensity of these discontinuities play an important role in controlling the engineering behaviour of rock masses. Joints, which are fractures in rocks along which there has been little or no movement, are the most common type of discontinuity found in all kinds of rock. They usually occur in sets, and as a rule, the joints in any given set are almost parallel to one another. It is commonly observed that rock masses are traversed by more than one set of joints. Typically at least three joint sets can be found, regardless of rock type or structural setting. Such a combination of intersecting joint sets splits the rock into small blocks.

The size and shape of these blocks will depend on the orientation of various sets as well as spacing between individual joints in each set, as sketched in Figure 3.1. Once the orientation and spacing distributions are known, estimates can be made of the volume of matrix blocks (Kazi and Sen, 1985). These blocks form the cornerstone of conceptualizing a jointed discontinuous rock mass. The size of matrix blocks is a useful index of fracturing intensity and can provide a basis for describing the quality of rock masses. Perhaps, more importantly, block volume or its inverse, volumetric frequency, is not directionally biased.

A similar approach has been suggested by Palmstrom (1985), who proposed a rock mass identification scheme based on block size, the compressive strength of the intact rock within individual blocks, and the volumetric joint count (an indirect measure of block size). This scheme is shown in Figure 3.2. However, neither block size or volumetric joint count provide an adequate description. Unfortunately, there is no unique relationship between the number of joints cutting a unit volume of rock and the number of matrix blocks obtained, due to the intersection of the various sets of joints.

Figure 3.3 a) illustrates two blocks of rock cut by a number of joints. It can be seen that for the same number of fractures, the number of matrix blocks produced in the two cases are different. Conversely, two blocks of identical volume, can have very different aspects, and hence the rock mass may have a very different response to mining, Figure 3.3b). Thus, it is not sufficient to measure volume only, and an additional factor is required to uniquely describe the matrix block. From examination of Figure 3.3 it is evident that if block shape can be measured a unique description of the block morphology can then be made. The effect of block shape is demonstrated in a relationship, shown in Figure 3.4, between block shape, volume and volumetric joint count.

If it can be assumed that the distance between individual joints within a given joint set is less than the length of those joints and that these joints are parallel, then the volume of the matrix block can be determined from the spacings as shown in Figure 3.5. Thus the matrix block is essentially a parallel piped. Even if four or more sets exist there will be a tendency for the typical, or average block, to be defined by the three most frequently (i.e. closest spaced) joint sets. On the other hand, the question may be raised as to the case of less than three joint sets. This situation has been found to be rare, and even though one or two joints may dominate a third set will in all probability exist, albeit at some high value for spacing.

Therefore, a knowledge of joint orientation and spacing will allow the size and the shape of matrix block to be defined. However, due to natural variations in these two parameters and the natural variation of joint length, completely joint bounded blocks may not always be formed. Therefore, there is a finite probability (or alternatively a finite number of blocks intersecting a given excavation surface) for intact rock to exist over a part of the matrix block surface. The presence of intact rock will tend to "key" the rock mass together.

The behaviour of the system of matrix blocks will not depend only on the size, shape, and amount of "keying" but also the strength of intact rock within the block, the strength along the different surfaces of the block and to a lesser extent the elastic properties of the intact rock (Young's Modulus and Poisson's ratio) and the elastic properties of the block's surfaces (normal and shear stiffness). These can be described in terms of the uniaxial compressive strength and the roughness of the joint surfaces. Therefore, parameters required include:

- number of discontinuity sets
- discontinuity orientation
- discontinuity spacing
- discontinuity length
- intact rock strength
- discontinuity surface.

Collection of the above parameters will allow the following design parameters to be estimated, namely,

- block size

- block shape
- "keying" factor
- block strength

The preceding discussion has outlined a conceptual approach to rock mass characterization, that of quantifying the matrix block. The succeeding discussion outlines aspects of measuring and quantifying the individual parameters required.

### 3.3.2 Number of Joint Sets

Both the mechanical behaviour and the appearance of a rock mass will be dominated by the number of sets of discontinuities that intersect one another. The mechanical behaviour is especially affected since the number of sets determines the extent to which the rock mass can deform without involving failure of the intact rock.

When considering the stability of a mine opening, a rock mass with three or more sets will generally constitute a three-dimensional block structure having considerably more "degrees of freedom" for deformation than a rock mass with less than three sets. For example, a strongly foliated phyllite with just one closely spaced joint set may give equally good tunneling conditions as a massive granite with three widely spaced joint sets. While the amount of overbreak in a drift will usually be strongly dependent on the number of sets.

The number of sets of discontinuities is an important parameter when evaluating hanging wall stability, though traditionally the orientation of discontinuities relative to the face is considered of primary importance. However, if insufficient sets exist, the probability of structurally controlled instability may be reduced almost to zero. On the other hand, a large number of sets having close spacing can influence the potential mode of stope failure, for example translational or buckling.

The number of sets will often be a function of the size of area mapped. In a preliminary investigation it is important to record all sets present. The recognition of individual sets will usually proceed simultaneously with orientation measurements. In some instances up to 150 joints may need to be measured, and the number of sets determined by contouring joint poles plotted on polar equal area nets.

If orientations are consistent, careful sampling may reduce the number of joints that have to be measured to define the number of sets. Once experience at a particular site has been gained, however, identification of joint sets can be accomplished simply by inspection.

### 3.3.3 Orientation

The orientation of a planar discontinuity is a unique measurement. Figure 3.6 shows how the orientation may be described by a vector. Orientation is usually measured with aid of a compass. The use of dip and dip direction is preferable to that of dip and strike, normally used by mine geologists, as it significantly reduces ambiguity and makes subsequent data manipulation more simple.

Where there is a strongly magnetic environment it will not be possible to use a compass to measure the dip direction (or strike) of the discontinuity. It is therefore recommended that a tape be used, stretched out parallel to the haulage, cross-cut or development and used as the basic azimuth against which orientations are recorded. The orientation of this tape is required in order that the true attitude of the discontinuity can be calculated.

Rock discontinuities are often not planar and the surfaces may be curved or irregular on a large or small scale. This factor causes inaccuracy in the determination of orientation.

A mean orientation can be established for a particular surface area by two methods:

1) A large number of individual measurements on a single surface may be located mathematically and a mean vector computed. This mean vector may then be regarded as the orientation of that discontinuity. Unfortunately, this method may require more than 50 measurements per surface.

2) The estimation of a mean plane may be done by eye. This is the most practical method, but can be of doubtful validity when only a small area of the joint surface is exposed.

The standard evaluation of discontinuity orientation measurement is by means of plotting the pole of each discontinuity surface on a stereonet. It is customary to state the frequency of joints of a certain orientation as the percentage of the total number of discontinuity orientations. A number of techniques are available for obtaining these frequencies (for discussion see Hoek and Bray, 1981). Once the percent frequencies are obtained the stereonet can be contoured on the basis of frequency and maxima identified. This procedure, though standard practice, is a non-parametric statistical process. No assumptions are made with regard to the underlying distribution of discontinuity clusters or sets. Little success, however, has been achieved in fitting primary distribution forms to orientation data (Einstein and Baecher, 1983). For underground mining studies it is adequate to identify maxima by simple inspection of the stereonet and assume that these maxima are representative of mean discontinuity orientations.

The question arises on the number of discontinuity orientations required before reasonable estimates of maxima and hence mean orientations can be made. From general principles of data evaluation, each unknown parameter should be supported by at least three measurements. Therefore to define the orientation of the three components of the matrix block (three directions, two unknown parameters with each direction) would require a minimum number of 18 discontinuity measurements at each locality. Kohlbeck and Scheidegger (1977) found empirically that 15-20 measurements at a locality, yield a consistent picture in most cases.

### 3.3.4 Spacing

Spacing is a measure of the distance between individual discontinuities within a rock mass or often more specifically between individual discontinuities within a joint set. Because spacing, or its inverse fracture frequency, can be used to estimate the number of joints, fractures and other planes of weakness that either intersect or are in close proximity to an excavation surface, it is perhaps the single most important characteristic of rock fabric. Discontinuity spacing has a direct influence on rock mass strength and behaviour; hence a direct influence on opening stability. Generally the closer discontinuities are to one another, the greater the degree of mining difficulty. The measurement and description of spacing is therefore of the utmost importance.

The terms used in the following discussions follow the definitions provided by Priest and Hudson (1981) and are as follows:

**Discontinuity Spacing, x**: The distance between an adjacent pair of discontinuities measured along a straight line of a given orientation within, or on the surface, of a rock mass. In general, the value varies with both scanline orientation and location. True spacing describes the perpendicular distance between adjacent discontinuities while the term apparent spacing is applied to all non-perpendicular distances, as illustrated in Figure 3.7.

**Mean Discontinuity Spacing, $\bar{x}$**: The mean value of spacings computed as:

$$\bar{x} = \sum_{i=1}^{n} x_i/n$$

where $x_i$ is the ith discontinuity spacing measurement along a scanline of length x yielding n values.

**Mean Discontinuity Frequency**: The mean number of discontinuities encountered per unit length along a scanline and hence the reciprocal of mean discontinuity spacing. The mean discontinuity frequency is a measure of the 'degree of brokenness' along a line for a given direction through a rock mass and applies for any discontinuity geometry. Thus, the mean spacing between discontinuities can be expressed as:

$$\bar{x} = 1/\lambda$$

where $\lambda$ is the fracture frequency.

Priest and Hudson (1983) have shown that the mean value for the true spacing between discontinuities can be estimated from the number of discontinuities (N) belonging to a specific discontinuity set intersected by a scanline of length (L) oriented in three dimensional space. The fracture frequency of discontinuities within the ith set is expressed as:

$$\lambda_{si} = N/L$$

The angle $\theta_i$ between a scanline of trend $\alpha$ and plunge $\beta$ and the normal to the ith discontinuity set (trend $\alpha_i$ and plunge $\beta_i$) is therefore given by

$$\begin{aligned}\cos\theta_i = {} & (\sin\alpha \cos\beta \sin\alpha_i \cos\beta_i) \\ & + (\cos\alpha \cos\beta \cos\alpha_i \cos\beta_i) \\ & + (\sin\beta \sin\beta_i)\end{aligned} \qquad (1)$$

or

$$\begin{aligned}\cos\theta_i = {} & \cos(\alpha - \alpha_i) \cos\beta \cos\beta_i \\ & + \sin\beta \sin\beta_i\end{aligned} \qquad (2)$$

If the discontinuity frequency along the ith set is $\lambda_i$ and there are N sets, the total discontinuity frequency along the scanline is found by summing the resolved frequencies from each set as follows:

$$\lambda_s = \sum_{i=1}^{n} \lambda_i \cos\theta_i \text{ for acute angles } \theta_i \qquad (3)$$

or

$$\lambda_s = \sum_{i=1}^{n} \lambda_i \left|\cos\theta_i\right| \text{ for all } \theta_i \qquad (4)$$

The true discontinuity spacing ($S_i$) for the ith set can therefore be expressed as:

$$S_i = 1/\lambda_i = 1/\lambda_{si}/\cos\phi_i \qquad (5)$$

The above sequence of calculations will allow the results of field surveys to be reduced and a mean spacing for each discontinuity set identified, to be determined. As the field measurements are done on "real" data the effects of impersistence (see Figure 3.8) of individual discontinuities are automatically catered for.

In nature joint spacing is a variable. Measurement (see, for example Einstein and Baecher, 1983) has indicated that this natural variation can be described by the negative exponential probability function which has the form:

$$f(x) = \lambda e^{\bar{\lambda}x}$$

where f(x) is the probability density function

$\bar{\lambda}$ is the mean discontinuity frequency

X is a spacing value.

The negative exponential distribution characterizes discontinuities spacings, and hence the location of the discontinuities themselves, as random occurrences in three dimensional space. Hudson and Priest (1983) present an interesting hypothesis for this apparent randomness in discontinuity location and apparent independence of individual discontinuities.

Consider a line through a rock mass intersecting discontinuities at a variety of points as illustrated in Figure 3.9a). Three possible probability density distributions are shown in the figure; the negative exponential distribution characterized by randomly distributed intersection points; the uniform distribution in which all spacing values up to a limiting value have an equal probability of occurrence; and, the normal distribution where the mean spacing is the most commonly occurring value. These three distributions cover three broad possibilities; a distribution with many small spacing values, one with equal numbers of spacing values and one with values equally distributed about a mean spacing.

According to Hudson and Priest (1983), the three distributions cover a wide range of possibilities and taken in combination are likely to approximate any in situ distribution. Since it is probable that a number of geological and mechanical factors combine during the various phases of fracturing to produce the current discontinuity patterns in rock masses, it may be assumed that a current distribution of discontinuity spacings has resulted from the superimposition of two or more component distributions. If this is the case, then the total discontinuity frequency is the sum of the component frequencies but the probability density of a given spacing value is not equal to the scaled sum of the contributions from the component distributions. This is because the spacing values have been superimposed and not simply added; small values have tended to survive but large values have tended to disappear.

This concept is illustrated in Figure 3.9b) and would appear to offer a reasonable explanation for observed distribution of discontinuity spacing.

Investigation into spacing distributions by Einstein and Baecher (1983) report that in only 8% of the spacing distributions analyzed, did the negative exponential model fail to satisfy goodness of fit criteria at the 5% confidence level. These results are a strong verification for the use of the negative exponential model to describe spacing distribution. An interesting observation made by Einstein and Baecher, is that while the average spacing varies with the orientation of the sampling line, exponentiality does not. Therefore, it is reasonable to establish relationships between average spacings using the simple trigonometric functions described earlier.

### 3.3.5 Rock Quality Designation (RQD)

The Rock Quality Designation (RQD) parameter was developed by Deere (1964) for quantitatively describing the nature of a rock mass from borehole core. As originally conceived it was intended as a modified form of core recovery, to be used as a semi-quantitative measure of rock quality. With time, however, it has gained substantial stature and forms an integral part of the most widely accepted rock mass classification methods.

RQD is defined as the percentage of all recovered intact core greater or equal to 0.1m, measured axially along the core between natural fractures, expressed with respect to the total length of the core run. RQD is therefore an indirect measure of discontinuity spacing. Figure 3.10 illustrates its measurement.

Though ostensibly a simple measurement, the use of RQD must be approached with caution. A number of significant errors can exist in its measurement. The first of these relates to the effect of core size.

As originally conceived, Deere used twice the core diameter of the standard NX core used in site investigation, to obtain the base measurement length of 0.1m. However, the standard exploration core, used in the mining industry, tends to be substantially smaller, typically between 20mm (EX) and 40mm (BX). Though obviously the same number of natural occurring fractures will be intersected, the small core diameter will tend to have many more drilling induced fractures, due to its lower strength. Heuze suggested that the base measurement length be reduced to reflect these effects in an attempt to ensure that the same value of RQD was recorded regardless of core size. Figure 3.11 gives the proposed relationship. For mine geologists faced with a myriad of production requirements the constant changing of base length for the different core sizes used is not appropriate. Using a relationship proposed by Priest and Hudson (1976) between RQD, fracture frequency ($\lambda$) and base measurement length ($\alpha$), a series of curves have been produced which allows the "corrected" RQD to be obtained from the "measured" RQD given the core size measurements were made on. These curves are given in Figure 3.12.

Doubt has been expressed by Priest and Hudson (1976) regarding the credibility of R.Q.D., as an index of discontinuity spacing. In many cases, it is believed that RQD is non-representative of the rock mass because it is strongly influenced by grouping of structures. Furthermore, it is insensitive to variations in rock quality when the mean discontinuity spacing exceeds 0.3m. Their recommendation was that different R.Q.D. thresholds be used for differing sizes of excavations.

Similar to discontinuity spacing, the value of RQD measured is very dependent on the direction of the borehole or sampling line. The effects of directional bias are particularly evident when comparing results obtained from underground for drilling as illustrated in Figure 3.13. Unlike discontinuity spacing, it is not possible to correct measured RQD for directional bias. If, however, corrected discontinuity spacing is available, a relationship between RQD and the inverse of spacing, fracture frequency can be used to recalculate RQD. This relationship, proposed by Priest and Hudson (1976) has the form:

$$RQD = 100\ \exp^{-0.1\lambda} \bullet (0.1\lambda + 1)$$

where $\lambda$ represents the overall fracture frequency along the particular sampling line selected (i.e. the combined frequencies of all the discontinuity sets that intersect the sampling line). This would however appear to be a somewhat fruitless exercise, using "real" data to derive an indirect measure of the "real" data.

Though the above criticisms exist and do not favour the use of R.Q.D. as a means of characterizing rock masses, a considerable amount of experience has been built up using RQD, particularly as a component of classification systems. It is believed that the use of discontinuity spacing and block volume will eventually replace RQD. However until such time, RQD will continue to be an important parameter in geotechnical engineering.

### 3.3.6 Persistence

The size of discontinuities can influence stability. Intuitively, the larger the size or persistence of discontinuities, the more detrimental may be their effect on opening stability. Unfortunately only the trace length of the discontinuity is visible on any rock exposure and often this length is censored by the limits of the exposure. This makes the estimation of size extremely difficult.

Figure 3.14 illustrates a scanline across a planar rock face, such as a drift wall, intersected by a set of discontinuities that, being of various limited extent, produce traces of various lengths. Because the scanline tends to preferentially intersect those discontinuities that may extend beyond the visible exposure, any measurement of trace length will be biased.

A substantial amount of research has been devoted toward the problem of resolving these biases and establishing a reliable estimator for trace length and for discontinuity size. Cruden (1977), Priest and Hudson (1981), LaPointe and Hudson (1981), and Worburton (1983) have all published work on the estimation of discontinuity trace length and size. Unfortunately, though a number of estimation methods exist, trace length distribution do not exhibit the consistent behaviour that spacings do. This makes the use of such methods difficult. Obviously it is possible for worst case scenarios to assume that all discontinuities are infinitely long. Experience has shown, however, that it is often adequate to rate the discontinuities present according to visual estimates of size, using simple descriptors as:

| | |
|---|---|
| • very low persistence | <1 m |
| • low persistence | 1-3m |
| • medium persistence | 3-10m |
| • high persistence | 10-20m |
| • very high persistence | >20m |

and simple sketches and block diagrams as illustrated in Figure 3.15.

Though this approach is subjective it is often sufficient to characterize the rock mass for design. In particular, if block volumes are to be calculated such data will give credence to the assumption that discontinuity length is greater than spacing.

### 3.3.7 Intact Rock Strength

A large number of methods are available for measuring the strength of rock. The most often used and the one that provides a basic means of classifying rock, is the uniaxial compressive strength test. This test is normally carried out in the laboratory using selected core samples.

Any ongoing data collection program tied into the mine's daily routine could involve a large number of observations in the rock mass. Obviously at each observation point it is not practical to take a rock sample for later laboratory strength tests. Some other indirect and rapid approach must be used. Two pieces of field equipment have been developed to assist in the rapid assessment of the uniaxial compressive strength of a rock. These are the point load strength apparatus and the Schmidt hammer. Descriptions on these apparatus and their use is given Brown (1981). Though both apparatus have their uses again because of the underground environment they are difficult to use. The Schmidt hammer in particular will give wrong estimates where there has been any loosening of the rock mass around a drift or mine opening.

Experience to date has shown that the best approach is to use a simple observational approach based on the guidelines given in Table 3.1. The limits of such an approach must be realized, however, and should always be carried out in conjunction with a comprehensive program of index strength of key rock units in the laboratory.

### 3.3.8 Discontinuity Strength

The profile and condition of discontinuity surfaces exert a major control on the shear strength and stiffness of the surface, and ultimately will influence the behaviour of the rock mass around an opening. A large amount of research has been directed toward characterizing discontinuity surfaces. The emphasis has been toward establishing methods of estimating shear strength along a surface. To date, the approach developed by Barton and Bandis (1982) has returned the most consistent results and is now widely used.

The relationship proposed by Barton and Bondis is based on three index parameters:

i) joint roughness coefficient JRC
ii) joint wall compressive strength JCS
iii) base friction angle, $\phi_b$

The equation relating these parameters is:

$$\phi_p = \mathrm{JRC} \log \frac{(\mathrm{JCS})}{(\sigma_n)} + \phi_b$$

where $\phi_p$ is the peak friction angle and $\sigma_n$ is the effective normal stress.

Both JRC and JCS can be obtained from field data while $\phi_b$ is determined from direct shear tests on saw cut samples.

## 3.4 Block Size and Shape

### 3.4.1 General

Block size is an extremely important indicator of rock mass behaviour. Block dimensions are determined by discontinuity spacing, by the number of sets, and by the persistence of the discontinuities delineating potential blocks.

The number of sets and their orientation determine the shape of the resulting blocks. These can take the approximate form of cubes, rhombohedrons, tetrahedrons, sheets, etc. Though, regular geometric shapes are the exception rather than the rule, since the joints in any one set are seldom consistently parallel, block shape can be reasonably well approximated by these forms. An interesting observation is that often block shape is consistent, even though other aspects such as volume may be significantly variable. Jointing in sedimentary rocks usually produces the most regular block shapes.

The combined properties of block size and inter-block shear strength determine the mechanical behaviour of the rock mass under given stress conditions. Rock masses composed of large blocks tend to be less deformable, and in the case of underground openings, develop favorable arching and interlocking. In the case of stopes, a small block size may cause the potential mode of failure to resemble that of soil, (i.e. circular rotational) instead of the translational, toppling, or blocky arch modes of failure usually associated with discontinuous rock masses. In exceptional cases "block" size may be so small that flow occurs, as with a "sugar-cube" shear zone in quartzite. This may be equivalent to caving ground. Thus, block size or perhaps more specifically, the projection of the block on the excavation surface and the size of the excavation itself are extremely important.

Blasting efficiency is likely to be largely a function of the natural in situ block size. It may be helpful to think in terms of a block size distribution for the rock mass, in much the same way that soils are categorized by a distribution of particle sizes as illustrated in Figure 3.16

Block size can be described either by means of the average dimension of typical blocks or by the total number of joints intersecting a unit volume of the rock mass (volumetric joint count $J_v$).

### 3.4.2 Volumetric Joint Count

A number of authors (e.g. Kazi and Sen, 1985; Palmstrom 1985) have proposed the use of the volumetric joint count as a means of characterizing a rock mass. Unlike line measurements, such as RQD or discontinuity spacing, the volumetric joint count ($J_v$) has the advantage of being free of directional bias.

The volumetric joint count is defined as the sum of the number of joints per cubic metre for each joint set present. Random discontinuities can be included, but will generally have little effect on the results.

Figure 3.5 shows a block diagram with three joint sets having spacings $S_1$, $S_2$, and $S_3$. The joint frequency for each set will be $1/S_1$, $1/S_2$, and $1/S_3$ respectively. The number of discontinuities intersecting one unit of rock mass, that is the volumetric count, is:

$$(Jv) = \frac{1}{S_1} + \frac{1}{S_2} + \frac{1}{S_3}$$

This definition of the volumetric joint count (Jv) takes into account in an unambiguous way all the joints occurring in the rock mass.

From observations of rock surfaces, the volumetric joint count can easily be calculated based upon joint spacings as shown above. The individual spacing for a joint set will normally vary within certain limits. By calculating the (Jv) from the closer and wider spacing for each set, the maximum and minimum degree of jointing can be found.

The volumetric joint count (Jv) was originally based upon mapping either in a tunnel/cavern or on the earth surface. In cases where no surface observations are available the (Jv) can be found using core observations either as RQD or as the joint frequency.

For the joint frequency $N_2$ (number of joints per meter borehole), Palmstrom (1985) has suggested that the transition from one-dimensional (in a borehole) to three-dimensional is:

$$(Jv) = K_2 \times N_2$$

The multiplying factor $K_2$ has been found empirically to range between 1.65 and 3.0 for equally distributed joints. The variation is caused by orientations of the borehole with respect to the joint sets. An average factor $K_2 = 2.0$ has been found to cover the most common jointing distribution. If the borehole is orientated parallel or perpendicular to a dominating joint set, the factor $K_2$ will have a greater range than indicated about and must be adjusted up or down respectively for the actual situation.

### 3.4.3 Block Size

A number of assumptions have been made in order to determine block size. These include:

1) Discontinuity spacing is less than discontinuity size; the side lengths of the block are therefore dependent on spacing values,

2) Discontinuities forming in the block are continuous over the entire area of a specific side.

3) That discontinuities occur in sets and that individual discontinuities within a specific set are to all intents and purposes parallel to each other.

4) That the block has a parallel piped form; that is the block is defined by three pairs of discontinuities taken from each of three discontinuity sets, respectively.

5) Where four or more discontinuity sets are present within the rock mass, the block is formed by discontinuities from the three most frequently occurring sets; that is those with the closest spacings.

Where the three defining discontinuity sets intersect each other at right angles, the block volume ($V_o$) is given by:

$$V_o = S_1 \times S_2 \times S_3$$

where $S_1$, $S_2$, and $S_3$ are the spacings between the pairs of discontinuities selected from the three discontinuity sets. If the discontinuity sets are not mutually perpendicular the volume must be adjusted:

$$V = V_o \times 1/\sin\alpha \times 1/\sin\beta \times 1/\sin\delta$$

where $\alpha$, $\beta$, and $\delta$ are the solid angles between discontinuity sets (see Figure 3.5).

Block size can also be estimated from the volumetric joint count. Figure 3.4 presents the relationship proposed by Palmstrom (1985). The influence of block shape on the block size should be noted. Kazi and Sen (1985) and Priest and Hudson (1976) have also proposed relationships between volumetric joint count and block volume as summarized in Figure 3.17. The differences reflected in the estimates are largely a function of the assumed distribution of discontinuity spacing. Palmstrom for example assumes that in nature discontinuities are uniformly spaced while Priest and Hudson use the negative exponential distribution to describe discontinuity spacing.

Though volumetric joint count may present a convenient shorthand method of estimating block size, particularly when data is relatively scarce, block volumes should be estimated directly from spacing data. As discussed in section 3.3.3 discontinuity spacing within a rock mass is not a constant value but is rather, naturally, variable. It follows then that block edge length must also be variable and, by association the block volume.

The effect of the underlying spacing distribution on the distribution of block volume has been investigated by Hudson and Priest (1979). Some of their results are given in Figure 3.18 and show clearly the influence of spacing distribution. Though normal and uniform distributions of discontinuities exist in nature, it has been found that by and large the negative exponential distribution represents the best approximation to observed data. Therefore it is reasonable to assume block edge length will follow the negative exponential distribution.

The resulting distribution of block volume is more difficult to assess. Though estimators for the mean and variance of volume of blocks formed by three mutually orthogonal discontinuity sets are available (see Miles, 1972), it has been found easier to construct volume distribution by use of Monte Carlo simulation. Randomly generated values of spacing are obtained from each of the three frequency distributions using the appropriate random number generator. The block volume is simply the product of these three values. By repeating the process a large number of times and sorting the results into assigned classes, the distribution of block volume is generated. A typical result is shown in Figure 3.19.

### 3.4.4 Block Shape

An important adjunct to the estimation of block volume discussed above is to obtain a measure of block shapes. Chappell (1987) has shown that block shape is important in assessing the re-distribution of stress that takes place when an excavation surface deforms.

In characterizing shape, it has again been assumed that the block is formed by three joint sets, thus is essentially a parallel piped. It is believed that this is a reasonable approximation. Even if more than four sets exist there will be a tendency for the typical block to be defined by the three most frequently (i.e. closest spaced) joint sets.

The proposed shape classification system is based on techniques used in sedimentology for the description of grain shape. Folk (1965) defines form as a measure of the relation between the three dimensions of an object. Thus particles (or rock blocks) may be classed quantitatively as compact (or equidimensional) elongated (or rodlike) and platy, with several intermediate categories. The so called 'form triangle' is shown on Figure 3.20. The triangle is made up of three axes, namely the ratio of S/L, $(L-I)/(L-S)$ and $S^2/LI(\Psi)$,

where S, I, and L are the measured edge lengths of the block. Any two of these axes can be used to define block shape.

## 3.5 Rock Mass Classification

### 3.5.1 General

The use of rock mass classification schemes has gained wide acceptance in recent years. These schemes seek to quantify those properties or features of the rock mass considered likely to influence its behaviour, and to combine these individual values into one overall classification rating for the rock mass. Empirical relationships have then been obtained by correlating the rock mass rating with observed conditions or behaviour. Such relationships, have with varying degrees of success, been proposed for a wide range of geotechnical problems; some of which are outlined in Table 3.2.

The following, paraphrased from Taylor (1976), outlines some of the objectives of a classification scheme:

- Divide a particular rock mass into groups of similar behaviour.
- Provide a basis for understanding the characteristics of each group.
- Facilitate the planning and the design of excavations in rock by yielding quantitative data required for the solution of real engineering problems.
- Provide a common basis for effective communication among all persons concerned.

These aims can be fulfilled by ensuring that the classification system has the following attributes:

- simple, easily remembered, and understandable
- each term clear and the terminology used widely acceptable
- only the most significant properties of the rock mass are included
- based on measurable parameters that can be determined by relevant tests quickly and cheaply in the field
- based on a rating system that can weigh the relative importance of the classification parameters
- functional by providing quantitative data for underground design

- general enough so that the same rock mass will possess the same basic classification regardless whether it is being used for a stope, drift, or shaft.

Though classification systems have been applied to a wide number of problems there are a number of potential shortfalls when such systems are used to predict mining conditions. The majority of existing systems have been developed from civil engineering experience and there are a number of deficiencies. These include:

- restrictions to levels of in-situ stress which are low compared with rock strength,
- too great a reliance on RQD,
- insufficient regard for hanging wall and footwall dips, foliation or shearing when assessing stope wall stability,
- too subjective an assessment of the affects of mining conditions (for example, in stress changes and blast damage) on stope stability.

Notwithstanding these deficiencies rock mass classification methods present an important method of describing rock mass quality and presenting a number which summarizes key geotechnical parameters.

Classification systems developed in recent years make use of measurable parameters, in an attempt to minimize judgmental bias. This corresponds to the recent trend towards more quantitative methods of rock mass description in general. The suggested method of rock mass description developed by the Working Party of the International Society for Rock Mechanics, for example, lists the following parameters:

- number of joint sets
- orientation
- spacing (block size)
- condition of joint walls (intact rock strength)
- condition of discontinuity filling
- joint roughness
- joint persistence
- joint opening (aperture)
- water seepage

All of these parameters can either be directly measured or scaled against carefully calibrated descriptive scales. Modern tunnelling classifications use similar parameters to develop an overall rating of the rock mass.

Though a large number of classification schemes exist (see Pakalnis, 1986 for a review) two have come to predominance. These are the Rock Mass Rating (RMR) proposed by Bieniawski (1973) and Q-system developed by Barton et al (1974). Components of the two systems are summarized in Table 3.3.

The Rock Mass Rating (Bieniawski, 1973) is based on applying a series of importance ratings to the various rock mass parameters. The overall Rock Mass Rating is given by addition of individual ratings. To obtain an estimate of likely ground conditions, this value is interpreted in terms of standup times for the particular dimensions of the opening under review. The relationship between Rock Mass Rating, span and standup time is shown in Figure 3.21a).

A similar concept is used in the Q-System (Barton, et al., 1974). In this scheme the structure of the rock mass, the frictional characteristics of fractures, and stresses surrounding the opening are assessed and combined to give the Tunnelling Quality Index (Q). This index has been related to the span of the opening that will stand unsupported as illustrated in Figure 3.21b).

The purpose of these classification systems is to finalize the empirical, judgmental approach towards tunnel design. As noted by Barton, the systems use the same thought process which would be utilized by a very experienced tunnel designer in assessing expected ground conditions and forming an opinion of the tunnelling procedure, based on an experience for a very large number of projects.

Over the years, since their initial inception, considerable experience has been gained in the use and application of these two classification systems. Because of significant differences in performance requirements, a number of modifications have been made to specifically address mining applications.

Laubscher (1977) modified Bieniawski's classification on the basis of experience gained in a number of caving operations. The classification, uses the same five rating parameters as Bieniawski's scheme but involves differences in detail. A second mining based classification scheme, based this time on Barton's Q rating scheme has been proposed by Mathews et al. (1981). Though the two mining classification systems use different methods for rating in-situ rock mass characteristics their final outcome has a similar format.

Geotechnical parameters, summarized in the form of a stability or rock mass number are plotted against a shape factor to assess empirically, the stability of surfaces bounding an open stope. The stabilit y number accounts for rock mass quality, the state of stress and the orientation of the exposed surfaces, while the stope factor accounts for the stope and size of the opening. On this basis, a high stability number and low shape factor should reflect stable exposures while the converse should reflect instability.

Laubscher's relationship is given in Figure 3.22 while Mathew's is given in Figure 3.23.

Details of the four classification schemes are given below.

### 3.5.2 Q-System

The Q-System is outlined in Table 3.3.

The physical significance of the Index is that the term $RQD/J_n$ represents in principle the average block size of the mass; $J_r/J_a$ the average total shearing strength of the joints and thus is a measure of the strength of the sides of a block; $J_w$ both the relative potential head of water on the support and the relative potential of the water eroding the joint infilling; and SRF both the structurally controlled kinematic possibility of dislodgement of blocks of ground and the concentrated stress relative confinement, fracturing, squeezing and swelling of the excavation walls.

- **Rock Quality Designation (RQD)**

RQD is a modified core recovery measurement. When borehole core is unavailable, RQD can be estimated from the number of joints per unit volume, where the number of joints per meter for each joint set are added together. A simple relation can be used to convert this number to RQD for the case of clay free rock masses (Hoek and Brown, 1980):

$RQD = 115 - 3.3^{*} JV$ (approximately)

where JV = Total number of joints per $m^3$
(RQD) = 100 for JV < 4.5)

This relationship is given in Table 3.4. Alternatively the graph shown in Figure 3.24 can be used.

It is important to realize the directional bias that is inherent within the measurement of RQD. This must be given careful consideration when assigning appropriate values (see section 3.3.4 for discussion).

- **Joint Set Number ($J_n$)**

The number of joint sets that can be observed in drift or pillar walls should be recorded. It is advisable to use all three walls (i.e. footwall, hangingwall, back) to reduce sampling bias. Only joints that can be traced for 1 m or more should be measured. Beware of blast induced fractures. Once the number of joint sets is determined, the appropriate joint set number is selected from Table 3.5. The average dip and dip direction of individual joint sets should also be recorded.

- **Joint Roughness Number ($J_r$) and Joint Alteration Number ($J_a$)**

These parameters should be relate to the weakest significant joint set or clay filled discontinuity in the given zone. The value of $J_r/J_a$ should in fact, relate to the surface most likely to allow failure to initiate. This ratio should be determined from inspection of the exposures underground. Judgment is required as to which is the most significant joint set in terms of potential stability problems. Values for $J_r$ are given in Table 3.6 and for $J_a$ in Table 3.7. The values given in Table 3.7 have been simplified from the original proposal of Barton.

- **Joint Water Reduction Factor ($J_w$)**

The joint water reduction factor allows for the water pressure on the joint walls, as well as the potential for the outwash and softening of joint gouge and values for $J_w$ are given in Table 3.8.

Due to the extensive and interconnected geometry of underground openings and often fractured nature of the rock mass immediately adjacent to these openings it is often assumed that the rock is in a drained condition, thus $J_w$ is often set to 1.

- **Stress Reduction Factor (SRF)**

Values for the stress reduction factor (SRF) are given in Table 3.9.

Kirsten (1983) has proposed a modification to Barton's proposed SRF parameter. This modification allows an initial prediction to be made on likely rock behaviour and replaces the SRF factor as follows:

$$SRF = 0.244\,K^{0.346}\,(H/S_c)^{1.322} + 0.176\,(S_c/H)^{1.413} \qquad (1)$$

A stress reduction limit is defined below which the SRF cannot fail; this is calculated as follows:

$$SRF = 1.809\,Q^{-0.329} \qquad (2)$$

where K is the ratio of the principal field stresses, H is the depth in metres which corresponds to the maximum principal stress, $S_c$ is the uniaxial compressive strength of the intact rock and Q is the rock quality based on the SRF calculated in equation 1. If the value of SRF calculated by equation 1 is lower than the SRF calculated by equation 2 then the rock mass will be controlled by the stresses and rock mass strength. Otherwise, the rock behaviour will be dominated by jointing.

This alternative method of assessing an SRF factor is shown in Figure 3.25.

### 3.5.3 RMR System

The RMR system is outlined in Table 3.3 while details of the ratings given to individual parameters are given in Table 3.10. This table is self explanatory. A number of notes on interpretation are however applicable.

- **Strength of intact material**

In absence of strength test results (either uniaxial compressive strength or point load strength index) the field method for estimating rock hardness outlined in section 3.3.6 can be used.

- **RQD**

See discussion in section 3.5.2.

- **Spacing**

Care must be taken in ascribing joint spacing due to sampling bias. The mean true spacing for each joint set should be determined and a rock mass average obtained. It is this value that is used to obtain the appropriate rating.

- **Joint Condition**

Joint condition should apply to average condition. Where several joint sets showing different surface properties occur it is suggested that the set that is considered to most influence stability be used.

### 3.5.4 Laubscher MRMR System

Laubscher defines a Mining Rock Mass Rating (MRMR) obtained from the summation of ratings for intact rock strength 0-20, RQD 0-15, joint spacing 0-25, and joint condition, 0-40, giving a total rating of 0-100. Descriptions and values associated with those basic ratings are given in Figure 3.26. Once an overall rating is obtained a series of adjustments are done. These adjustments cater for the effects of:
- weathering
- geologic structure in a stope or drift back and walls
- blasting.

These adjustments allow the in-situ rock mass rating to be obtained.

The determination of an overall rating is relatively straightforward. The following notes apply.

- **Joint Spacing (See Figure 3.27)**

Provision is made in Figure 3.27 for obtaining the joint spacing rating for one, two or three joint systems. In the case of four or more joint sets then the three most frequently occurring joint sets are used (i.e. the closest spaced).

- **Joint Condition and Groundwater**

The rating is given in Figure 3.28. Consideration is given to discontinuity waviness (Parameter A), roughness (Parameter B), alteration (Parameter C), and infill (Parameter D), under four different groundwater conditions.

The condition rating is then a combined percentage of the total possible of 40, that is,

Joint condition rating = 40 x A x B x C

- **Adjustments to MRMR**

Weathering: for rocks that deteriorate readily during the life of an opening, adjust RQD rating by 95%, UCS rating by 96%, and joint condition by 82%, giving up to 75% adjustment to MRMR.

Joint Orientation: the orientation of joint sets will define those rock blocks that may kinematically slide or fallout from an excavation surface. The adjustments, simplified and slightly modified from Laubscher's original proposal are shown in Figure 3.29. These are made to the joint spacing rating.

### 3.5.5 Mathews System

Mathews defines a stability number (N) expressed as:

N = Q' A.B.C. where,
Q' is the modified NGI Rock Mass rating
A is a stress factor
B is a rock defect orientation factor
C is a design surface orientation factor.

The stability number has been empirically related to the hydraulic radius as shown in Figure 3.23.

Within the stability number Mathews et al (1981) have attempted to account for three types of failure. These are:

(a) Structurally controlled failures caused by opening and movement along unfavorably oriented structure (usually zones of low stress).

(b) Stress controlled failure through intact rock.

(c) A combination of structural and stress controlled failure of the bridge of intact rock between structure.

- **Modified NGI Rock Mass Rating (Q')**

The rock mass quality index Q (see section 3.5.2) is modified by setting the stress reduction factor (SRF) to 1. All the other factors are unchanged, hence the modified index Q' accounts for rock mass strength and structure only.

- **Rock Stress Factor (A)**

The rock stress factor (A) replaces the stress reduction factor (SRF) to more accurately reflect stresses acting on exposed surfaces of open stopes at depth. This factor is the ratio of intact rock strength to induced stress where:

(a) intact rock strength is defined as the uniaxial compressive strength, and

(b) induced stress is defined as the stress acting parallel to the exposed stope wall or roof under analysis.

Bridging type failure between fractures should not occur when the ratio of intact rock strength to induced stress exceeds 10. Any failure under these conditions should be related to movement on defined structure only and for this case, the rock stress factor (A) is set to 1.0.

Failure will be stress controlled as the ratio of intact rock strength to induced stress approaches unity. For this case, the rock stress factor (A) approaches zero.

Combined stress and structurally controlled failure can occur between these extremes and a straightline relationship has been assumed to determine the factor A. From experience, the ratio of intact rock strength to induced stress should not be less than 2 as indicated in Figure 3.30 and this relationship can be used to flag potential problems.

- **Rock Defect Orientation Factor (B)**

This factor is included to account for the presence of persistent structure paralleling or intersecting exposed surfaces, Figure 3.31. The value of the factor is 0.5 for structure paralleling the exposed surface and 1.0 for structure intersecting the exposed surface at right angles. The factor is selected as follows:

(a) Determine the orientation of the most persistent set of fractures based on relative spacing and continuity.

(b) Determine the angle of intersection with the exposed surface under consideration.

(c) Refer to Figure 3.31 and select the orientation factor (B). Stereographic plots can be used to determine the relative angle between joint sets and design surfaces.

- **Orientation of Design Surface Factor (C)**

Stope backs or roofs are inherently less stable than sidewalls because of the influence of gravity. Barton et al (1974) suggest that rock quality in a wall is hypothetically improved 5 times compared to a roof. They also recommend a ESR of 1.6 for permanent mine openings. However, some block fallout and spalling can be tolerated in a "non-entry" type of excavation. Therefore, it seems conservative to suggest that a vertical wall should be at least 5 x 1.6 = 8 times as stable as a horizontal roof. If the factor (C) for a horizontal surface is set to 1, then its value will increase as the angle of dip increases. The maximum value is 8 when the surface is vertical. The formula used to obtain the relationship between factor (C) and the angle of dip is given below and graphed in Figure 3.32.

Factor C = 8 - 7 cosine (angle of dip)

The factor accounts for the effect of gravity and should be applied to roofs, hanging walls and possibly steep footwalls when adverse structure has been identified.

### 3.5.6 Correlation between Rock Mass Classification Systems

The two base classification systems, that is Bieniawski's RMR method and the Q system of Barton, use similar parameters, though with different values, to rate the rock mass. It is, therefore, not unexpected that there would be a good correlation between the systems. This aspect has been investigated by Bieniawski (1985) and by Laubscher (1987). Their findings are summarized in Figure 3.33.

## 4.0 DATA COLLECTION PROCEDURES

## 4.1 Geological Data

Rock masses and their component discontinuities can be described by three principal methods:

(a) exposure description
(b) drill hole and drill core description
(c) photography.

### 4.1.1 Exposure Description

A number of methods can be used to describe an exposed rock face or drift wall. The simplest is a qualitative approach using standard descriptors. This approach perhaps represents the most widely used method of describing a rock mass. Because the approach is essentially qualitative it is limited in its direct usefulness and is probably best applied during early stages of an investigation where general trends are being studied.

A large body of literature exists on different terminologies, and the numerical bounds of these, that can be used to describe rock masses. The most comprehensive set of descriptors is contained within a set of guidelines produced by the International Society of Rock Mechanics. A summary is given in Table 4.1 of the various parameters that can be observed on an exposed face and the associated descriptions that apply. Though this simple form of rock mass characterization has gained wide acceptance, it is not recommended for mining ground control studies. More quantitative measures are required. Two sources from which to obtain the required information is from rockmass classification or from scanline surveys.

- **Rock Mass Classification**

As discussed in section 3, a number of classification systems can be used to rate the quality of the rock mass. The schemes all use similar parameters, though weighted and combined in different ways, to obtain measures of rock quality. Because of these similarities, underground data collection should be directed towards collecting information in such a manner that it can be used as input into all four classification schemes. A common data collection sheet was therefore developed. This sheet is given in Figure 4.1, while the proposed recording procedure is detailed in Appendix A.

- **Scan Line Surveys**

The use of rock mass classification methods as a means of collecting relevant data serves a useful "shorthand" approach. Often, however, more detailed information is required, particularly with respect to the orientation, spacing and continuity. Ideally any procedure developed to record such data should be such that the work can be incorporated within the daily routine of mine staff.

A number of methods are available for detailed data gathering of discontinuity information. Most of these were originally developed for open pit investigations where access to different rock exposures is relatively easy and where generally good visibility exists. Unfortunately underground access is often difficult, generally being limited to a drift wall or back, and visibility is extremely poor.

At the Thompson Mine, the first attempts at detailed data collection made use of detail line survey techniques. This method, described in Appendix B, was found to be extremely time consuming and though the information collected was of high quality, a substantial amount of the data collected was not particularly useful.

Experience at the mine has indicated that discontinuity orientations, roughness and condition, is relatively constant throughout. Spacing was, however, variable and was believed to be the single most important geologic parameter, apart from rock type. Data collection procedures were therefore modified in order that emphasis could be given to spacing. Rather than precisely measure the location of each discontinuity that intersected the sampling line (or cell) it was decided to simply count the number of discontinuities, belonging to an identifiable discontinuity set, that intersected the sampling line. Knowing the number of discontinuities, their mean orientation, the length of the sampling line and the orientation of the sampling line, it is a trivial exercise to determine the mean spacing for each discontinuity set identified. Details of the procedure are given in Appendix C.

In developing the above data collection methods a number of problems were encountered. These relate to various types of sampling biases. Errors in sampling arise from three sources: sampling error, estimation (statistical) error, and measurement error. Sampling error is caused by plans that are not representative, estimation error is caused by statistical fluctuations from one sample to another, measurement error is caused by inaccuracies in the way individual elements are measured. Specifically at Thompson the most significant bias is believed to be due to sampling errors, primarily as a result of:

- **Directional Bias**: The number of intersections that a reference sampling line make with a set of discontinuities is dependent on the attitude of the discontinuities and the orientation of the sampling line. Where the solid angle between the two is 90 degrees (i.e. the sampling line is perpendicular to the joint set) the maximum number of intersections is obtained, thus true spacing, fracture frequency and RQD can be measured directly. As the angle between the two decreases the number of intersections is reduced, ultimately to a minimum when the reference line parallels the strike of the discontinuity set. The result is to significantly bias spacing, fracture frequency, and RQD measurements. This concept is illustrated in Figure 4.2.

  To correct for this type of bias it is normal practice to use three independent sampling lines at right angles to each other. Unfortunately this is not always possible in a mine where haulage, and access development parallel each other and the orebody. To minimize directional bias, adjustments can be made to the collected data.

Two options for adjusting the data exist. The first is the Terzaghi correction factor (Terzaghi, 1965) which is applied to orientation data in order to adjust the number of discontinuities intersected by the sampling line for directional bias; while the second is a simple 3-D geometrical adjustment to spacing measurements to obtain an estimate of true spacing. These two adjustments are available through the two data handling computer programs STERINCO and SCANLINE. Details of the two programs are given in Appendix F.

- Operator Bias: Operator bias can occur during detail line surveys. A major problem is that of separating blast induced fracturing from natural discontinuities. The only solution believed feasible is the use of judgement. This comes with experience .

- Size Bias: A number of errors exist that can be related to size of the discontinuities being measured; these included proportional length bias, censoring bias and truncation bias. Einstein and Baecher (1983) present a discussion of these biases. Though it is recognized that such biases do occur, it is believed for the practical purpose of data collection for stope design they are not of particular significance.

### 4.1.2 Geotechnical Core Logging

- General

Diamond core logging is one of the most expensive forms of geological data collection, and generally forms the major expenditure in an exploration or mine development program. In consequence, it is essential that the maximum of data is obtained from the drill core.

Besides providing geological and ore grade data, drill core can be a valuable source of geotechnical and engineering information pertaining to rock conditions at depth. This can form the basis for subsequent design. It is therefore recommended that any recovered drill core be logged for engineering, as well as geological and grade parameters. The amount and type of geotechnical detail recorded will vary depending on the project and the individual hole. As a result, data recording can vary from a summary log indicating the basic generalities of the rock mass characteristics to a detailed log of the orientation and nature of every discontinuity.

Problems encountered in the drilling of a hole frequently tell more about the ground conditions than can be obtained from logging of the recovered core. Thus, a geotechnical logging method should contain, where appropriate, the following information in addition to a record of geological features.

A summary of the drilling events should be noted. This summary enables a reconstruction of events, allowing an engineer to see exactly what occurred in the drilling of the hole and may also aid in the interpretation of core condition. Items such as the rods becoming stuck or hard to turn; the need for bit or core catcher replacement; deviations in usual drilling techniques etc., all add to this documentation and serve to produce a log that can be used in the interpretation of the ground conditions.

- Logging of Exploration Core

The major difficulty with exploration drill core is that it is usually of small size (AX or EX) and poor quality in relation to the measurements required. The main object of exploration drilling is to recover a sample of the rock at minimum cost. Therefore, it is generally only worthwhile extracting very broad information from this core.

Unfortunately exploration core is the standard type of core found in most mining operations. It is therefore appropriate to carry out summary engineering logging. This method can provide fairly detailed engineering data from drill core at the same time as geological logging is performed without an appreciable decrease in the rate of logging.

The principal features that can be recorded during summary logging are:

(a) **Core Recovery**

- Total Core Recovery
- RQD

Though core logging standards state that mechanically induced fractures should not be included, experience at Thompson has found that the selection on what is mechanically induced and what is not varies considerably from operator to operator. To minimize such bias it was decided that all fractures, that is both naturally occurring and mechanically induced, should be measured during logging. Though this will result in higher frequencies, spacings, RQD etc., measurement will be consistent thus ultimately allowing viable comparisons.

(b) **Discontinuity Properties**

Discontinuity Properties can be recorded under defined classes of dip; dip being measured relatively to the core axis (see typical logging sheet, Figure 4.3).

- **Frequency**: The number of discontinuities occurring the sampling interval should be recorded. Where appropriate, the number of discontinuities occurring within the defined dip classes can be recorded.

- **Roughness**: Due to the small size of exploration core, it is not possible to record roughness in detail. The schemes outlined in Figures 4.4 and 4.5 are considered appropriate.

- **Condition**: Again due to the small size of exploration core, and to drilling losses it is often difficult to describe condition.

- **Type**: The type of discontinuity recorded (e.g. foliation, fault joint etc.).

(c) **Rock Type**

- **Type**: The generic name of the rock.

- **Hardness**: An indirect measure of the uniaxial compressive strength of the rock. Guidelines are given in Appendix C.

- **Alteration/Weathering**: Where pervasive change in the rock fabric has occurred it should be systematically described.

An alternative approach is to use the parameters given in the two main rock mass classification systems (see section 3.5.1).

Appendix D presents the standard core logging procedures adopted at the Thompson Mine.

Similar to applying the results of scan line surveys the user must be aware of the various errors and biases that may occur during core logging. Again the direction of the borehole probably represents the most serious bias, though operator bias, particularly where core is poorly handled, can also be significant.

## 4.2 Photographic Techniques

### 4.2.1 General

Photographs are an invaluable aid to the interpretation of geological and geotechnical data. A clear scale should always be included in the picture, since photography of the geology or ground conditions will be carried out at varying levels of detail e.g. core, mapping cells, drifts or stopes. Though colour slides or prints present a more recognizable picture of the subject being photographed it has been found that black and white prints often form a useful adjunct, particularly when scale measurements are done.

Photographs should be indexed and filed. When required, slides can be projected on a screen or alternatively examined with a hand lens or a light table. For scale measurements, the photographs can be projected onto the rear of a ground glass sheet, e.g. through a light table, and the size adjusted to natural scale, see Figure 4.6.

### 4.2.2 Underground Photography

A difficult set of conditions are encountered when photographing exposures underground. Often, the drift walls are covered in dust or mud and, prior to mapping or photography, an attempt must be made to clean off the rock. For drifts less than about 5 m wide, a short focal length lens should be used, e.g. 28mm. In this case, it should be borne in mind that most standard flash units will not illuminate as wide an angle as the lens will cover and suitable adaptions, like slave flash units, may be needed. The wall should be marked to show the area mapped, its location, and sometimes each discontinuity mapped.

One of the most successful techniques for underground photography is to mount the camera on a tripod and then use a flash a number of times to evenly illuminate the face from different positions. Experience will give indications as to the correct exposure setting.

### 4.2.3 Core Photography

Photographs of drill core provide a permanent record of the core condition upon recovery, and hence can be an invaluable aid to the interpretation of ground conditions, and therefore to mine planning. The photographs show detail which cannot be easily recorded on drill logs, and they can be filed for easy reference by both geologists and engineers. Many mines find that the photographs are extremely useful for interpreting economic geology, as well as structural geology, particularly if the core is to be split for assay.

All core photographs should include ample, clear identification of the hole number, footage and core direction (way up). In addition it is important that core be photographed dry. Where possible core should not be split until the film has been developed and satisfactory photographs obtained.

A 28 mm, wide-angle lens has proved to be most convenient when photographing core. A suitable frame can be constructed of dimensions to suit the size of the core boxes and the width to height ratio of the 35 mm photograph. A title block should be included to indicate workplace/stope number, drill hole number and footage interval. A metre scale should also be included.

Subdued daylight provides the best lighting for core photography, and direct sunlight should not be used. Flash or photofloods may be needed in adverse natural lighting conditions, in which case different film requirements must be observed with regard to flash or floodlamp type. Regardless of the lighting system used, consistency is of the utmost importance for interpretation of the core photographs, as similar colours or shades will be recorded differently in varying lighting conditions.

## 5.0 MINING CONDITIONS

### 5.1 General

The previous section has outlined methods of geologic data collection. By and large such data reflects the distribution of geologic and geotechnical properties before significant mining has taken place.

As mining progresses, significant re-distribution of stress occur. This re-distribution leads to zones of stress concentration and zones of stress relaxation. At some point the stress re-distribution may result in sufficient concentration to induce rock failure; alternatively relaxation may reduce confining stresses and clamping forces to a level that allows fallout, block sliding or other gravity controlled mechanisms.

The occurrence underground of stress induced failure or fallout is marked by changes in the need for rock reinforcement, the amount of dilution and recovery within stopes, and ultimately, mining cost. These aspects of changing mine conditions are all measureable by systematic observation and inspection; that is, without instrumentation. Such visual evidence can be recorded as follows:

A. Changes in the rock mass surrounding the stope:

- stress fracturing at drift corners
- slabbing of drift walls
- loosening and movement along fractures

B. Changes in rock surfaces within stope:

- slabbing or wedging of back
- buckling or loosening of hanging wall as indicated by large waste at drawpoint
- working of drawpoint brow or pillar

C. Changes in the condition of blastholes:

- stress failure in the blasthole walls producing ovality
- shear movement along pre-existing planes.

D. Amount and type of support installed and any resupport.

To monitor such changes, underground areas should be visited regularly and a survey made of conditions. These will change as mining progresses. By tracking the amount and type of change which occurs, data is obtained on the response of the rock mass to mining. Ultimately this data can be used to calibrate the various empirical, analytical and numerical models used for prediction purposes. Before discussing the monitoring process it is instructive to briefly review the effect of stress around a mine opening.

## 5.2 Stresses around Underground Openings

### 5.2.1 General

The stresses which exist in an undisturbed rock mass are related to the weight of the overlying strata and the geological history of the rock mass. This stress field is disturbed by the creation of an underground excavation and, in some cases, this disturbance induces stresses which are high enough to exceed the strength of the rock. In these instances failure of the rock adjacent to the excavation boundary can lead to instability which may take the form of gradual closure of the excavation, roof falls and slabbing of side walls, or in extreme case, rock bursts.

### 5.2.2 Stress Distribution

The rock pressure acting on underground opening is a result of three factors:

1. Vertical load due to the weight of the overlying rock;
2. Horizontal load due to geologic forces;
3. Adjacent mining activity.

With depth, therefore, there is an increase in both vertical and horizontal loads.

A method of representing the stress field around openings is by means of principal stress trajectories. These are imaginary lines which indicate the direction of maximum load (principal stress). Figure 5.1 shows a typical stress distribution before mining and the re-distribution that occurs as a result of mining. Where the stress lines crowd together there is a build-up in compressive loads, while where the stress lines part, there is a decrease in load resulting in stress relief. In this figure it was assumed that the vertical loads were greater than horizontal loads thus resulting in increased loads on the sidewalls and relaxation of the back. If the reverse were true, that is horizontal loads were greater than vertical loads, increased loads would be found in the back and relaxation of the side wall would occur.

It must be emphasized that, due to geologic history and the effects of adjacent mining, the stress lines do not necessarily flow in the vertical and horizontal directions as shown in Figure 5.1. For example, a footwall haulage will experience changing stress or load conditions and directions as stoping approaches and passes it. Such a possible change is illustrated in Figure 5.2.

### 5.2.3 Failure

Changes in stress or load can lead to ground problems.

Rock is generally weak in tension due to the presence of various natural discontinuities such as joints or foliation surfaces. Therefore, where the rock is in a loosened or relaxed state, the attitude of these planes will control stability. The type of failures that can be experienced include, ravelling, wedges sliding from its sidewalls and fall-off from the back. Figure 5.3 illustrates this concept. In contrast where high loads are experienced failure is typified by slabbing of walls, horizontal fracturing of backs or crushed "hour glassing" of pillars. These are illustrated in Figure 5.4.

## 5.3 Monitoring

### 5.3.1 General

An underground mine consists of a complex geometry of haulages, drifts and stopes. Each is mined at a different period of time, thus creating a constantly changing induced stress environment. This changing environment results in changing rock mass behaviour, which is particularly evident as stress approaches, and exceeds peak strength. In these instances new fractures are formed and existing fractures extended, thus irrevocably altering the rock mass characteristics. Hence, it is not sufficient to only record the in-situ geological properties prior to mining, but it is also necessary to continuously monitor changing conditions during mining.

A monitoring scheme should provide systematic information about the type, location, severity and rate of ground control problems or movements. Such a data base will provide a detailed history of stoping operations and ensure a method for providing advance warning of impending failure.

The primary objectives of monitoring are:

- to maintain safe operational procedures for the protection of mine and equipment;
- to provide advance notice of ground control problems, thus allowing for the possible modification of mine plans;
- to provide geotechnical information useful to analyze failure mechanisms and for conducting future ground control studies.

A number of different monitoring systems are available. These include:

1. Visual Monitoring

    - **"section sketching"**; visual monitoring of drift and stope conditions and change in conditions as stopes advance;
    - **"hole survey"**; blasthole, bolthole or monitoring holes are checked on a regular basis for signs of stress failure, crumbling and shear as indicators of stress and disturbance;

2. **Instrumentation**

- **"Instrumentation"**; installation of measuring points, extensometers convergence meters at selected locations.

The use of visual monitoring procedures is discussed below.

### 5.3.2 Visual Monitoring

- **Section Sketching**

All active underground areas should be visited on a regular basis to record ground conditions. These changes should be monitored using various measures of mining difficulty. Observations that can be made include:

a) **Stress State**: The general conditions of the haulage ramp, drift or stope being inspected, can be described using the five-fold classification system outlined in Table 5.1.

b) **Type of Failure**: The type of failure that occurs can be recorded according to the nine simplified failure modes illustrated in Figure 5.5.

c) **Scaling**: The amount of effort required to scale the area can be recorded using the three-fold classification described in Table 5.2.

d) **Support Type**: A record of installed support can be maintained using the support categories outlined in Table 5.3. Where appropriate the support dimensions should be given (eg. size and spacing of bolts).

e) **Support Load**: The condition of the support system can be monitored with respect to load. Table 5.4 describes a classification that could be used. When recording load condition the percentage falling within each category should be noted.

f) **Stope Stability**: A record can be maintained of mining efficiencies and production statistics for each stope. This record will provide information on tons blasted, ore recovered, dilution experienced, and volume of backfill placed.

A typical recording sheet is given in Figure 5.6.

In addition to systematic observation, the use of underground photography is highly recommended. Photographs of specific events provide a valuable permanent and visible record.

- **Blasthole Survey**

Blasthole performance is often a reliable indicator of changing conditions. Blastholes should therefore be monitored on a regular basis. Two pieces of information can be recorded, namely blasthole condition and number of redrills in a specific area.

Blasthole condition can be recorded using the classification system shown in Figure 5.7, while redrills can be categorized according to the following three-fold system; Class 1: 0% to 10% redrills; Class 2: 10% to 50% redrills; and Class 3: 50% to 100% redrills. A typical recording sheet is shown in Figure 5.8.

- Drawpoint Monitoring

Systematic observation of the type, size and shape of the muck that appears at a drawpoint can provide information on stope stability. Such a record will also give data with regard to the success of primary blasting procedures.

- Reporting

a) **Inspection:** The monitoring programme described above should be carried out on a regular basis. The time interval between inspections will depend on how quickly change is occurring. Initially all active areas should be inspected monthly. Where warranted (i.e. those areas with rapidly changing conditions) the interval between inspections should be reduced.

b) **Responsibility:** The inspections should be carried out by a nominated person reporting to the Mine Superintendent. After each inspection a summary report should be prepared. To assist in reporting, a mine longitudinal showing ground condition in a colour coded form should be maintained. This longitudinal should be updated after every inspection.

## 6.0 DATA COLLECTION AND REDUCTION AT THOMPSON MINE

### 6.1 General

Detailed collection for rock mass characterization and on ground condition was initiated at the Thompson Mine during 1984 as part of a major rock engineering study into crown pillar removal. Data from this program forms the cornerstone to the rock engineering data base at the mine. This data base has been expanded during the current study in order to characterize the ore, immediate hanging wall and immediate footwall rocks over the entire mine. Though underground data collection is still underway sufficient information is available to establish mine averages. These averages can then be used to determine if a specific area represents poorer or better quality ground than would typically be expected.

The following sections give a brief description of the work being carried out, the process of data reduction, and the resulting mine average criteria.

### 6.2 Work being done for Rock Mass Characterization

#### 6.2.1 Engineering Geological Core Logging

In addition to standard geologic logging of Primary Development (PD) core, mine staff are recording a number of geotechnical parameters. These parameters are based on the input required to assess rock quality (Q) as defined by the NGI rock mass classification methods, Barton (1974). Data recording is restricted to the orebody, the immediate hanging wall and the immediate footwall.

Typically EX or AX core barrels are used in PD drilling giving rise to approximately 20mm and 30mm diameter core respectively. These small sizes make it difficult to realistically describe parameters such as joint roughness or condition. It was therefore decided to default values for these paramet s to mine average. Only where substantial clay or other strength reducing factor was observed, would the default values be overridden.

Information from PD drilling is recorded and stored on Inco's mainframe data retrieval system. This allows mine wide manipulation of borehole data. A typical output from the system is given in Figure 6.1; note the rock quality data contained with the description columns. This data can be individually retrieved and an average estimate of Barton's Q rating obtained, see Figure 6.2. The data manipulation shown in this figure was carried out with the aid of a LOTUS 123 spreadsheet program.

PD core is not retained once it has been logged. Therefore, to provide a permanent visual record and to allow subsequent re-examination of critical areas, the core is photographed using 35mm colour slide film. A typical result is shown in Figure 6.3.

### 6.2.2 Discontinuity Mapping

Information on discontinuity geometry was obtained initially from detail line surveys. However, this approach was found to be extremely time consuming due in part to the inherent difficulties of mapping underground and in part due to the plethora of information collected during this type of survey.

Experience had indicated that the most important fabric parameters were: the number of discontinuity sets present, the mean orientation of the discontinuities within these sets, the spacing or frequency of discontinuities within a set and their surface condition. Therefore, a more streamlined method of data gathering was developed. This method, described in Appendix C, allows the requisite information to be obtained very rapidly allowing the process to be incorporated into a normal underground inspection. A typical data collection sheet is shown in Figure 6.4.

## 6.3 Data Reduction

### 6.3.1 General

There are three distinct areas of data reduction and compilation. The first deals with overall mine structure, and orebody geometry; that is aspects such as orebody width, orebody dip, hanging wall rock type, hanging wall thickness etc. The second and third areas of interest deals with rock fabric and rock mass quality.

### 6.3.2 Mine Structure

Information in mine structure is obtained from the results of geologic mapping and PD based mine data retrieval system. The data can then be manipulated and output obtained in a variety of forms, including point and contoured data. This approach to data reduction is the subject of a separate report and therefore only a brief mention is given here.

Typical results from the 1C orebody are given in Figures 6.5 to 6.10. These figures show the distribution of orebody dip, orebody width, hanging wall rock type, footwall rock type, immediate hanging wall thickness and immediate footwall thickness, respectively, using either contours or symbols. An overlay giving mine geometry is given in Figure 6.11.

### 6.3.3 Fabric

To assist in evaluating the data collected from the various line surveys, a fabric data handling program STERINCO has been developed. The program has the following functions:

- extract fabric data from Lotus 123 files
- ensure orientation data is in dip and dip direction format (as opposed to strike and dip)
- perform Terzaghi correction for directional sampling bias
- plot pole points of discontinuities on lower hemisphere stereoplot
- construct contoured stereoplot of pole points.

A typical pole point plot and contoured stereoplot output from the program is shown in Figure 6.12. In addition to the above feature the program will also sort data according to user specified criteria; for example, by rock type, discontinuity type, discontinuity orientation etc. A description of the program and documentation for its use are given in Appendix F.

The program STERINCO varies from other stereo programs in the respect that each pole is assigned a probability distribution. For the contouring routine, this converts the pole to a standard distribution which will compensate for probable field measurement errors.

The distribution used is a spherical normal or Fisher distribution, see Figure 6.13. This results in the assumed probability of error tapering off to zero within 10 degrees in any direction. The formula describing this distribution:

$$\frac{K}{4\pi \sin h\,(K)}\, e^{K \cos\beta}$$

where, $\beta$ is the spherical angle between the recorded pole and the point on the distribution, and K is the dispersion factor which is set at 242.

The process used for data reduction can be summarized as follows:

1) Correct raw data from detail line survey for directional bias. Use the Terzaghi correction factor if considered appropriate.

2) Plot dip and dip direction data on stereonet. Define the number of joint sets and their attitudes.

3) Sort data according to defined joint sets.

4) Obtain histograms and statistical summaries of discontinuity length, frequency roughness, and

waviness for each joint set defined.

5) Use mean spacing of the three most frequently occurring joint sets to obtain shape factor.

6) Assume spacing and discontinuity frequency follows negative exponential distribution. Use Monte Carlo Simulation to generate distribution of block volume.

7) Assess rock mass quality using Q, MRMR and N systems.

## 6.4 Thompson Mine Averages

### 6.4.1 Orientation

From the joint mapping and detail line surveys that were carried out, no significantly different structural domains have identified. Similar joint orientations, though not intensities, appear to be pervasive through all rock types in which measurements were done. This trend has been noted throughout the mine. All dip directions and dip angles were then plotted on a single stereonet, shown in Figure 6.14.

Four major joint sets can be seen on the stereonet. These are summarized in Table 6.1. The jointing is predominantly high angle. Joint set 1 represents foliation surfaces. Joint sets 2 and 3 are approximately perpendicular to each other. Comparison with regional trends would appear to indicate that the orientation of joint set 2 is parallel with that of the MacKenzie dyke swarm. Joint set 4 is low angled and is generally perpendicular to the foliation surfaces.

Statistical contouring of the stereonet suppresses the orientation of individual surfaces. As a number of these represent minor faults or shear zones, important to stability, separate stereonets are constructed. Experience to date has found that the majority of shear zones are associated with joint sets 1 (foliation) and 2. It is therefore reasonable to conclude that the orientations of minor faults and shear zones are adequately described by the composite stereoplot, Figure 6.14.

### 6.4.2 Condition

Geological mapping has recorded the condition of the different major joints and shear sets associated with the overall jointing pattern. Generally, such zones are widely spaced and only locally are they concentrated to the extent that they could be called a zone of weakness.

The mapping revealed that bedding slips commonly occur in the schists and in the quartzites, parallel to the foliation surfaces, joint set 1. Some may be only graphitic planes or consist of wider zones of graphitic bearing or biotite rich schists. Others may have fillings up to 15 to 30 mm of mud, chlorite, carbonate, quartz, pyrite or gouge. Within the skarns, fillings are carbonate, and the joints have "healed". Another feature encountered is the interformational contact slip. These are particularly evident in amphibolites, and may have selvages of biotite up to 100 and 200 mm thick at their contacts. Some of these bedding slip faults are water bearing.

The majority of joints in Sets 2 and 3 are tight. A few (approx. 10%) were encountered with a filling up to 30 mm of quartz, carbonate, chlorite or mud and on occasion were water bearing. The shallow dipping joint set (joint set 4) consists of tight joints, with or without, alteration along the surface. Occasionally, they may have an infilling of chlorite or carbonate, generally 1 mm thick and rarely up to 1 cm in thickness. These joints are normally widely spaced but occasionally (such as in the mylonitized quartzite zone in the 310 Fresh Air Drift) they occur at a spacing of 150 mm.

### 6.4.3 Continuity

Fabric continuity can rarely be measured directly in the field, as only traces of individual surfaces are generally visible. Some estimation method must be adopted. Statistical techniques exist (Worburton, 1980) but are complex and not considered pertinent to this level of study. A method based on classification of fabric origin, actual trace length observed and prior experience has therefore been devised.

A four-fold classification of fabric origin is proposed. In order of decreasing size, this is:

- Foliation surfaces (joint set 1), minor faults and shear zones.
- Joints orientated parallel to the MacKenzie dyke swarm (joint set 2).
- Joints orientated perpendicular to the MacKenzie dykes warm (joint set 3).
- Flat lying joints normal to foliation surfaces (joint set 4).

Each joint set encountered during the detailed mapping was examined and assigned to one of the above classes. Based on actual observation of trace lengths within the individual classes and prior experience, an estimate of average discontinuity length was assumed. These are given in Table 6.2. Discontinuity lengths, however, are not found as single unique values. They are best represented by a probability distribution. Recent research (Baecher and Einstein, 1977) has found that the negative exponential distribution best fits discontinuity data. This distribution has the advantage that the standard deviation has the same value as the mean. Thus, once the mean is known (or assumed) the entire distribution can be constructed. Figure 6.15 shows distribution for the four estimated mean lengths in Table 6.2.

An approximate statistical check has been made on the above empirical estimate of joint trace length. Miller and Barret (1979) have reported that the population mean trace length ($\mu$) can be estimated from:

$$\mu = (1 + \sigma_t - \sigma_c)h/(1 - \sigma_t + \sigma_c)$$

where,

$$\sigma_t = \frac{\text{No. of Joints Transecting the Exposure}}{\text{No. of Joints Intersecting the Exposure}}$$

$$\sigma_c = \frac{\text{No. of Joints Wholly Contained in the Exposure}}{\text{No. of Joints Intersecting the Exposure}}$$

and,

$h$ = Height of the Exposure

These equations apply to vertical discontinuities that strike normal to the exposure. Values for t and c were obtained from measurements done during detail line surveys. Estimates of mean length were made for joint sets 2 and 3. Joint set 1 generally ran parallel to the survey line and length data could not be analyzed using the above equation. The results obtained gave an estimate for joint trace length for joint set 2 of 4.9m and for joint set 3 of 4.5m. These values compared well with estimated trace lengths given in Table 6.2.

### 6.4.4 Spacing

The frequency of the different fabric elements was studied in two ways. The first used the measurements of RQD carried out by the mine as part of their development drilling program. The second was a more detailed approach using the data generated from detail line surveys.

Since 1980, the mine has been recording the RQD during logging of exploration and P.D. drill core. To retrieve this data a computer program 'BHARQD' was developed. Histograms of the distributions of RQD values in each of the major rock types were constructed. These are shown in Figures 6.16 to 6.18. Table 6.3 gives a statistical summary of the histograms.

Priest and Hudson (1981) have proposed an approximate relationship between RQD and the joint spacing. It has the form:

$$S = \frac{(110.4 - RQD)^{-1}}{3.68}$$

This equation has been used to estimate mean joint spacing, Table 6.4.

The values reported in Table 6.4 represent the average spacing between all joints belonging to specific joint sets. When evaluating rock mass characteristics such as rock mass strength, it is appropriate to use overall values. However, for stability studies, where orientation is important, it is necessary to evaluate spacing between individual joint sets. This was the basis for the second study. The scanline and detail line survey records were reviewed to determine the number of intersections made with each joint set. This value was used to calculate mean joint spacing. As the survey lines were generally not perpendicular to the trend of the jointing, an orientation correction was made. The results are given in Table 6.5. Spacings are variable. For example, they ranged from 50mm to 1m for joints in quartzite belonging to the foliation set (joint set 1). Often clustering was observed.

### 6.4.5 Classification of Rock Units

The determination of rock mass quality at Thompson has largely been carried out using Barton's Q system. Though other systems, for example Laubscher's MRMR method and Mathew's N method, are being evaluated as aids to stope design, the Q system has remained the cornerstone for assessing variability in rock mass quality.

The various parameters required for the Q system have been described in detail in section 3.5.2. Values for RQD were obtained from the RQD data base outlined above. Appropriate values for Jn, Jr and Ja were selected after examination of drill core, review of the results of detail line surveys and from specific observations underground. As the mine was generally dry with only a few minor wet zones; Jw was set at 1.

The results of rock mass classification are given in Table 6.6. These represent the estimated Q-Values for each major rock type, for the mine as a whole; that is, the values are averages. At specific localities the individual parameters and the estimated Q-Value can be significantly higher or lower. As RQD has the greatest influence on the Q-Value, ranges of RQD measured for each of the rock types, have been used to investigate the overall variability of Q. These are also reported in Table 6.6.

From the values of Q given in Table 6.6, the different rock types can be grouped with respect to their rock mass quality and anticipated engineering behaviour. Four categories are recognized. These are:

- **Class I** - Biotite Gneiss, Skarn (unleached), Amphibolite, Hanging Wall Quartzite and Footwall Quartzite

- **Class IIA** - Hanging Wall Schist

- **Class IIB** - Ore Rocks

- **Class III** - Footwall Schist

- **Class IV** - Skarn (Leached)

The rocks contained within each class are expected, broadly, to have the same engineering properties such as strength and deformation. A sub-division of Class II has been made to emphasize the more anistropic characteristics of the schist; a result of the presence of large, continuous foliation surfaces.

## 7.0 CONCLUSIONS AND RECOMMENDATIONS

As a first step toward predicting stability, it is necessary to systematically describe the rock mass in which mining will take place. Three steps are involved:

1) the identification of significant properties
2) measuring the geological parameters that control these properties
3) compilation of the measurements into a form suitable for design.

Studies at the Thompson Mine have indicated that the single most important geological parameter, apart from rock type, is discontinuity spacing; particularly the spacing of the discontinuity set (foliation) that parallels the orebody. Observations have shown that where spacing is small there is an increased incidence of ground control problems. Data collection procedures have therefore been directed toward obtaining information that will allow spacing to be adequately characterized. Initially, standard data collection methods developed for open pit studies were used. These were found to be untenable in the underground environment, largely due to time requirements. Ultimately, it was found that a simplified version of scan line surveys provided the most appropriate method of data collection.

Because spacing is a three dimensional property its value is dependent on the direction of the line in space along which measurements are made. Though corrections are made to obtain true spacing the use of block volume proved to be a useful unbiased descriptor. Block volume is derived from the matrix block bounded by discontinuities from the three most frequently discontinuity sets. By taking due cognizance of the inherent natural variability of discontinuity spacing, the distribution of block volume can be estimated. This distribution is akin to the particle size curves that form the basic classification system in soil mechanics. By also calculating the shape factor for these matrix blocks a relatively unique characterization of the discontinuities within a rock mass can be obtained.

It is believed that this approach offers a number of advantages over existing methods and will provide a useful addendum to existing rock mass classification techniques. Ultimately the determination of block volume and shape could be used to construct a model of mine openings as an aid to stability evaluations. Further research is required to develop this approach, particularly with the view to determining relationships between block size and shape, and types of failure. If such a relationship could be established it would greatly assist in the prediction of stope and drift stability.

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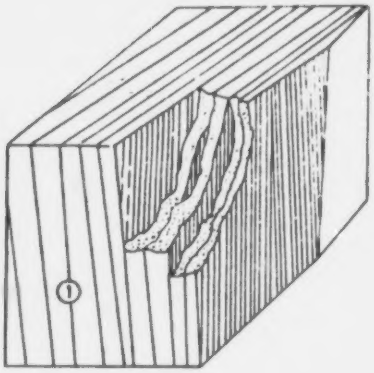
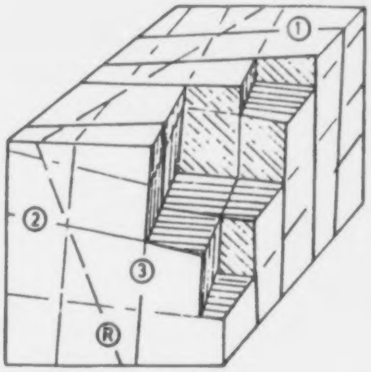
# EXAMPLES THAT DEMONSTRATE THE EFFECT OF THE NUMBER OF JOINT SET ON THE MECHANICAL BEHAVIOUR AND APPEARANCE OF A ROCK MASS

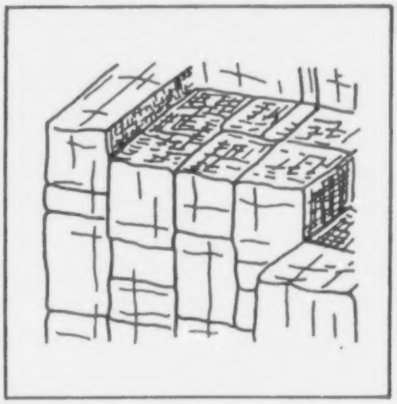
Figure **3.1**



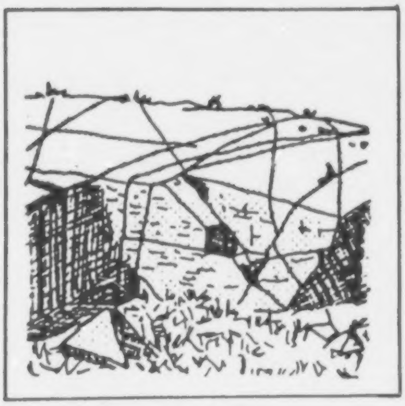
One Joint Set



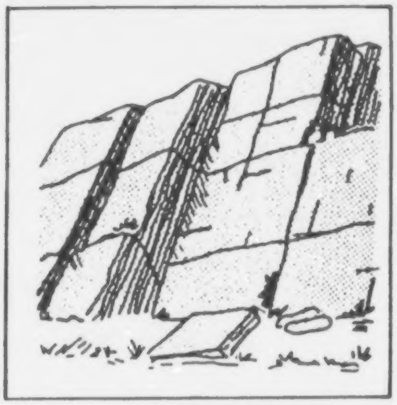
Three Joint Sets



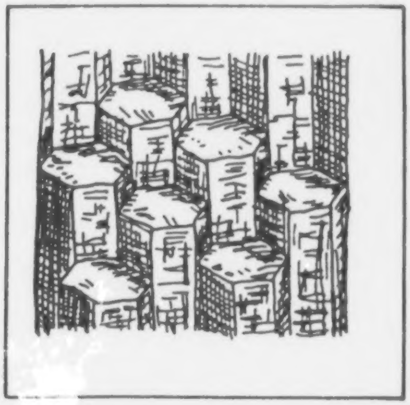
Blocky



Irregular



Tabular

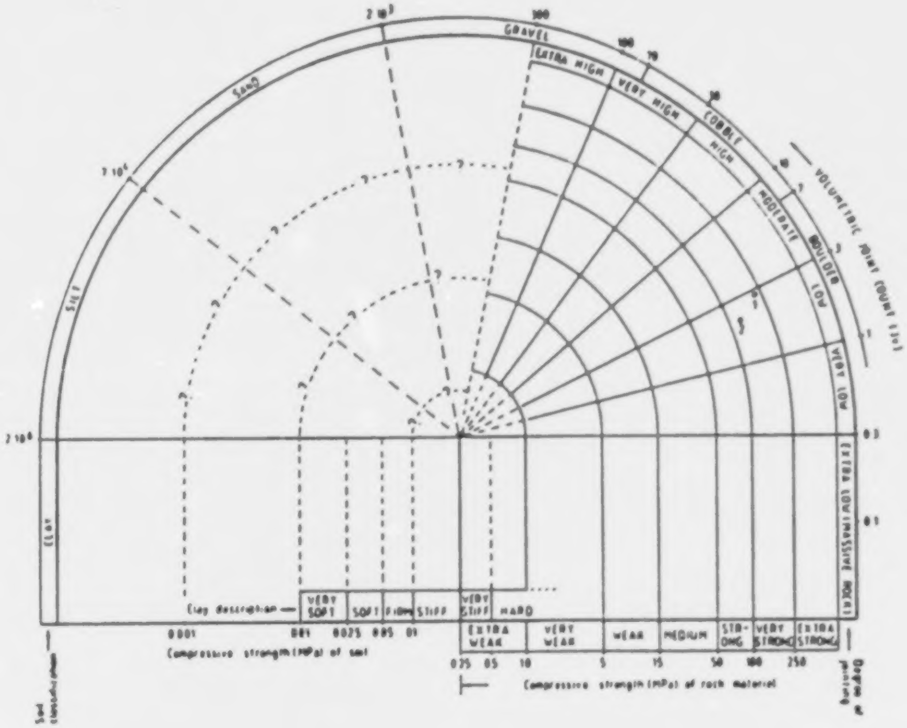


Columnar

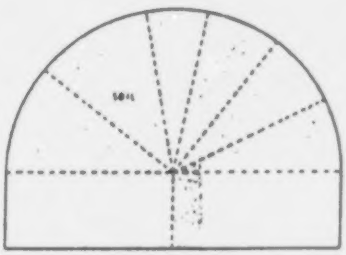
PROJECT NO.......... DRAWN.......... REVIEWED.......... DATE..........

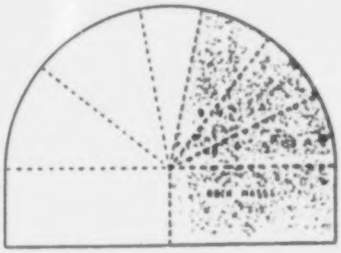
# IDENTIFICATION OF SOIL AND ROCK MASSES FROM PARTICLE/BLOCK SIZE AND COMPRESSIVE STRENGTH

**Figure 3.2**



SAND
SILT
GRAVEL
COBBLE
BOULDER
CLAY
EXTRA HIGH
VERY HIGH
HIGH
MODERATE
LOW
VERY LOW
EXTRA LOW (MASSIVE ROCK)
VOLUMETRIC JOINT COUNT (Jv)
Clay description
VERY SOFT
SOFT
FIRM
STIFF
VERY STIFF
HARD
EXTRA WEAK
VERY WEAK
WEAK
MEDIUM
STRONG
VERY STRONG
EXTRA STRONG
0.001
0.01
0.025
0.05
0.1
Compressive strength (MPa) of soil
0.25
0.5
1.0
5
15
50
100
250
Compressive strength (MPa) of rock material
Soil classification
Degree of jointing
0.3
0.1



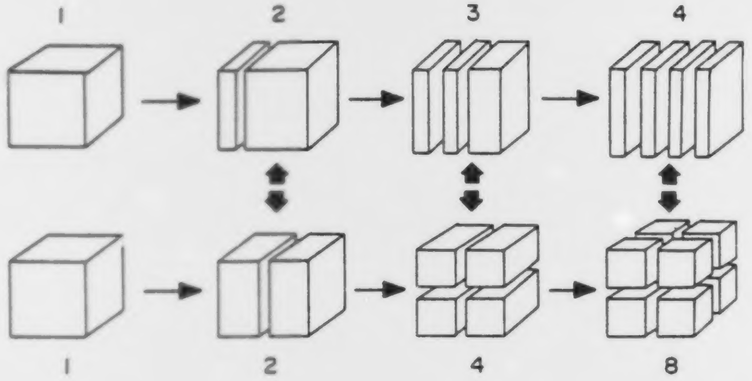


ROCK MASSES

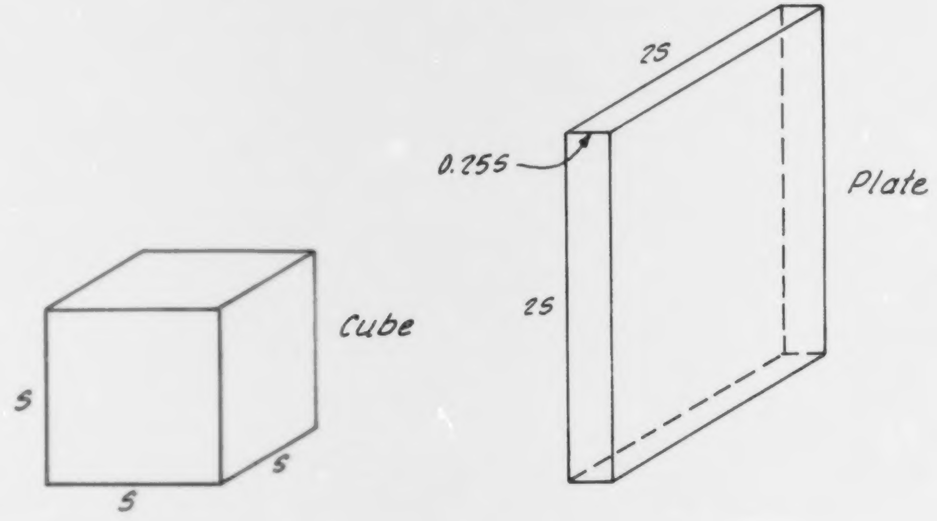
PROJECT NO. ............ DRAWN ............ REVIEWED ............ DATE ............

## BLOCK SHAPE

Figure **3.3**

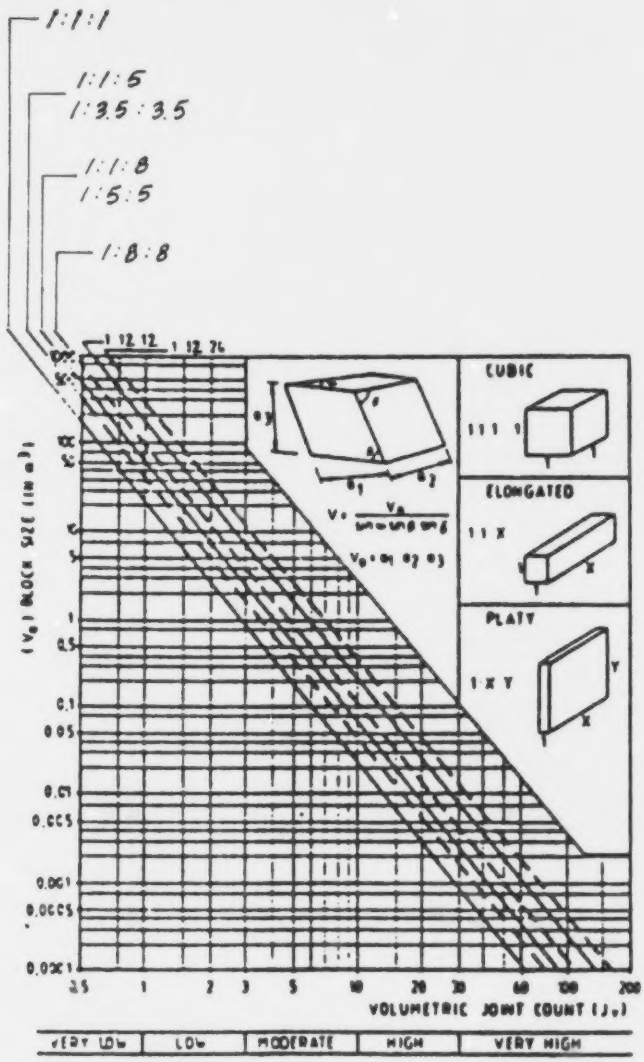


a) Same number joints - different block size.


2S
0.25S
Plate
2S
Cube
S
S
S

b) Same block size and same number of joints, different aspect.

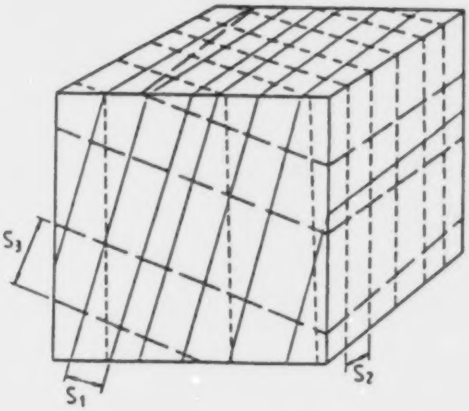
| INTER BLOCK SIZE AS A FUNCTION OF (Jv) AND JOINT PATTERN | Figure **3.4** |
|---|---|

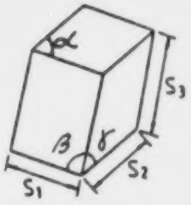


(After Palmstrom, 1985)

## DETERMINATION OF BLOCK VOLUME

Figure **3.5**




$\alpha$
$S_3$
$\beta$
$\gamma$
$S_2$
$S_1$

UNIT BLOCK

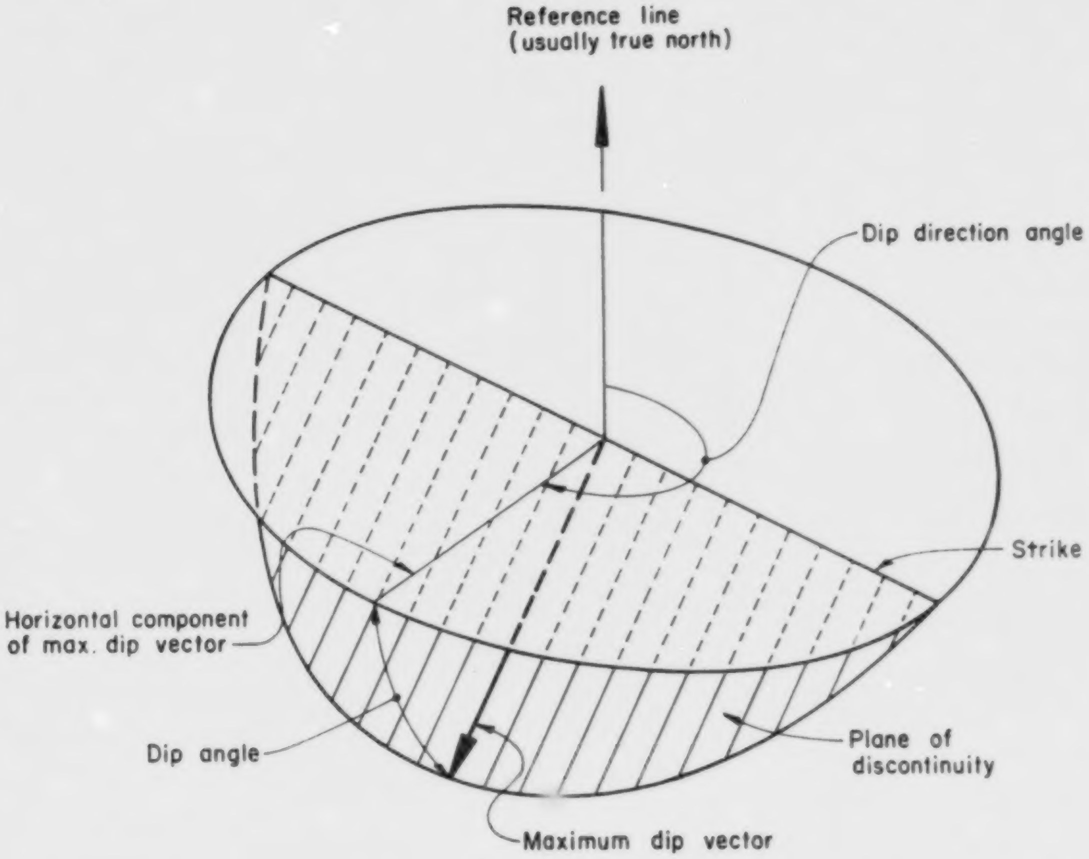
$$V_0 = S1 \cdot S2 \cdot S3$$

$$V = V_0 \cdot \frac{1}{\sin\alpha} \cdot \frac{1}{\sin\beta} \cdot \frac{1}{\sin\gamma}$$

PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

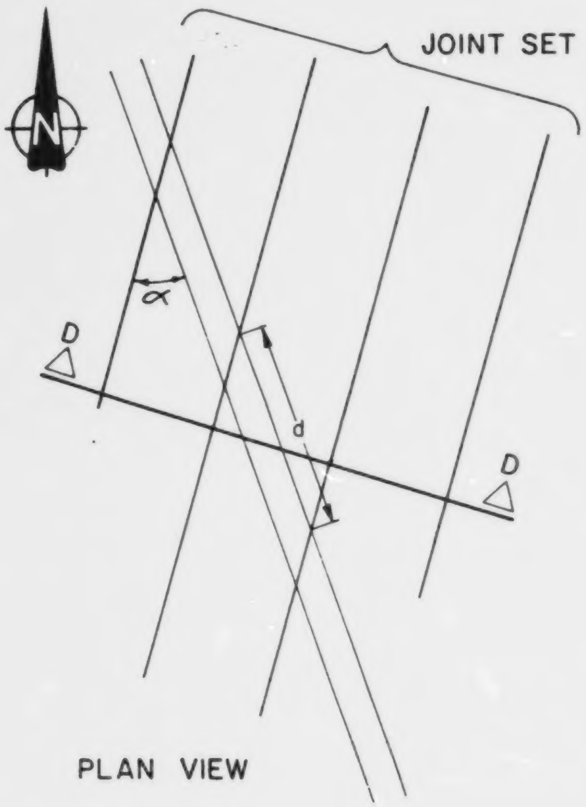
## ORIENTATION OF A DISCONTINUITY

Figure **3.6**

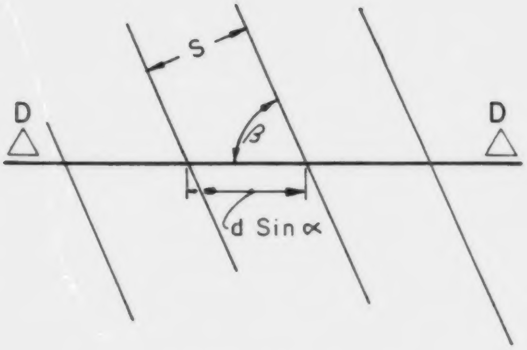

Reference line
(usually true north)
Dip direction angle
Strike
Horizontal component
of max. dip vector
Dip angle
Plane of
discontinuity
Maximum dip vector

# TRUE AND APPARENT SPACING BETWEEN JOINTS OF THE SAME SET

Figure **3.7**



PLAN VIEW



SECTION D - D

LEGEND

| | |
|---|---|
| d | Apparent spacing |
| α | Angle between strike of joint and adit direction. |
| β | True dip of joint |
| S | True spacing $d \sin\alpha \sin\beta$ |

EXAMPLE

Joint set dipping 61°/105°
Adit running
$d = 50$ m
$\alpha = 36°$
$\beta = 61°$
$S = 50 . \sin 36° . \sin 61°$
$= 25$ m

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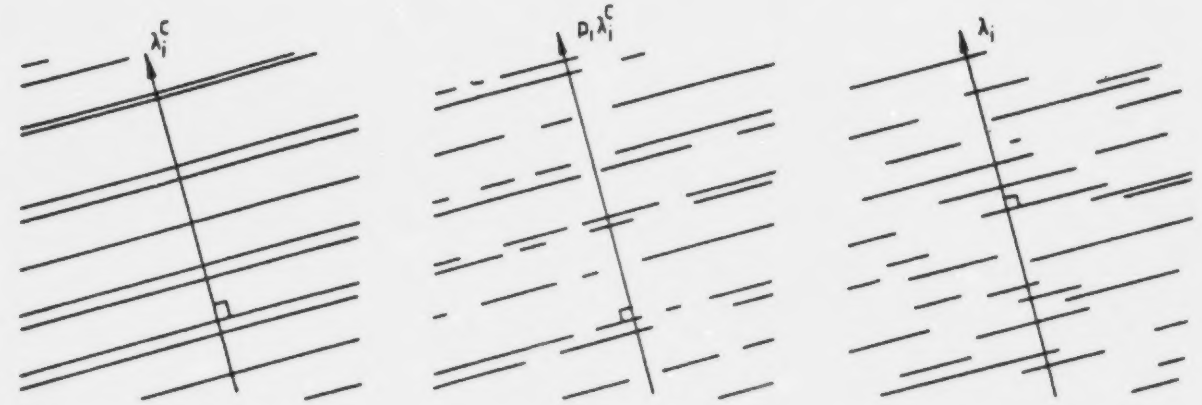
# EFFECTS OF IMPERSISTENCE ON DISCONTINUITY FREQUENCY

Figure **3.8**



(a) Persistent discontinuities

(b) Impersistence produced by intermittent discontinuities

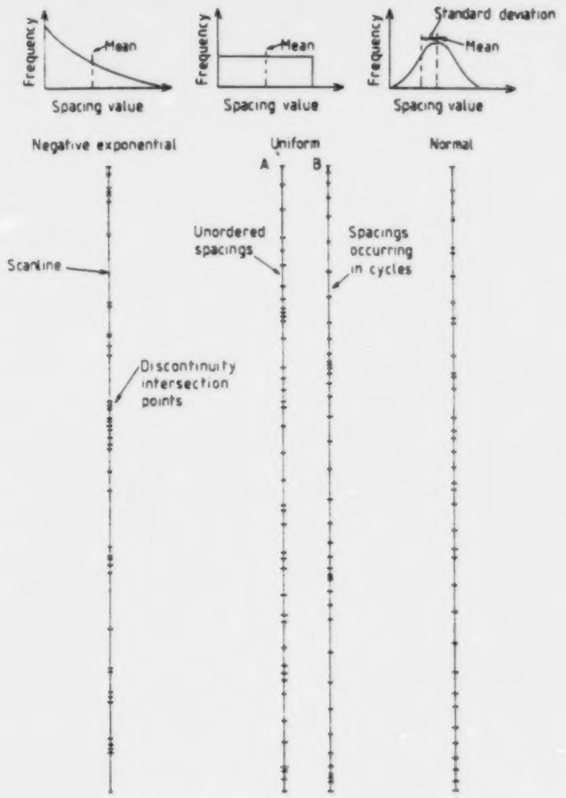
(a) Separate impersistent discontinuities

Traces produced by sets of persistent and impersistent discontinuities intersecting a plane surface. (a) Persistent discontinuity planes. (b) Intermittent discontinuity planes. (c) Separate impersistent discontinuity planes.

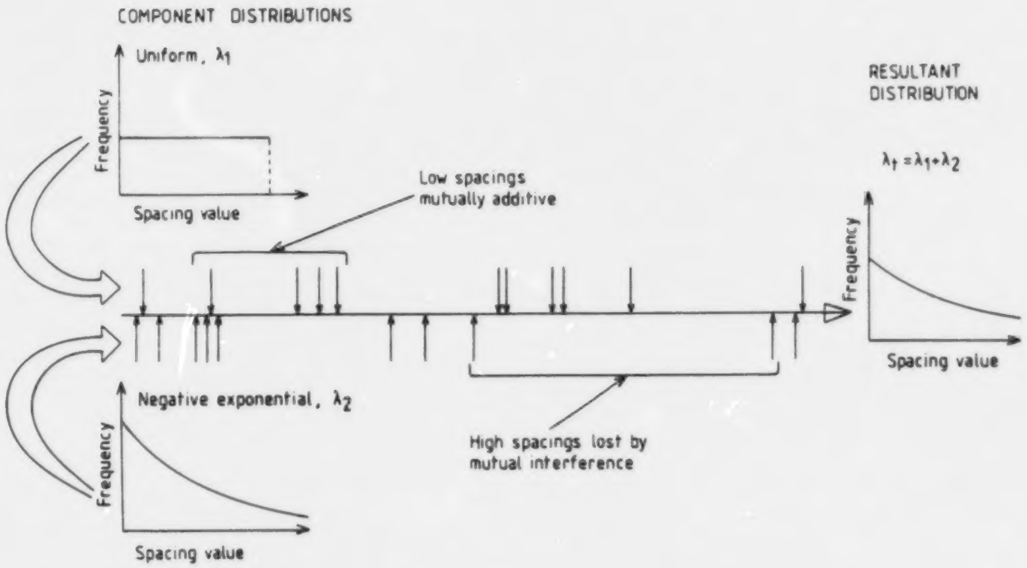
PROJECT NO. .......... DRAWN .......... REVIEWED .......... DATE ..........

# DISTRIBUTION OF DISCONTINUITY SPACING

**Figure 3.9**


Frequency
Mean
Spacing value
Frequency
Mean
Spacing value
Standard deviation
Frequency
Mean
Spacing value
Negative exponential
Uniform
A
B
Normal
Unordered spacings
Spacings occurring in cycles
Scanline
Discontinuity intersection points

**a)** Examples of theoretical discontinuity spacing distributions.


COMPONENT DISTRIBUTIONS
Uniform, $\lambda_1$
Frequency
Spacing value
Low spacings mutually additive
RESULTANT DISTRIBUTION
$\lambda_t = \lambda_1 + \lambda_2$
Frequency
Spacing value
Negative exponential, $\lambda_2$
Frequency
Spacing value
High spacings lost by mutual interference

**b)** Combination of two discontinuity spacing distributions.

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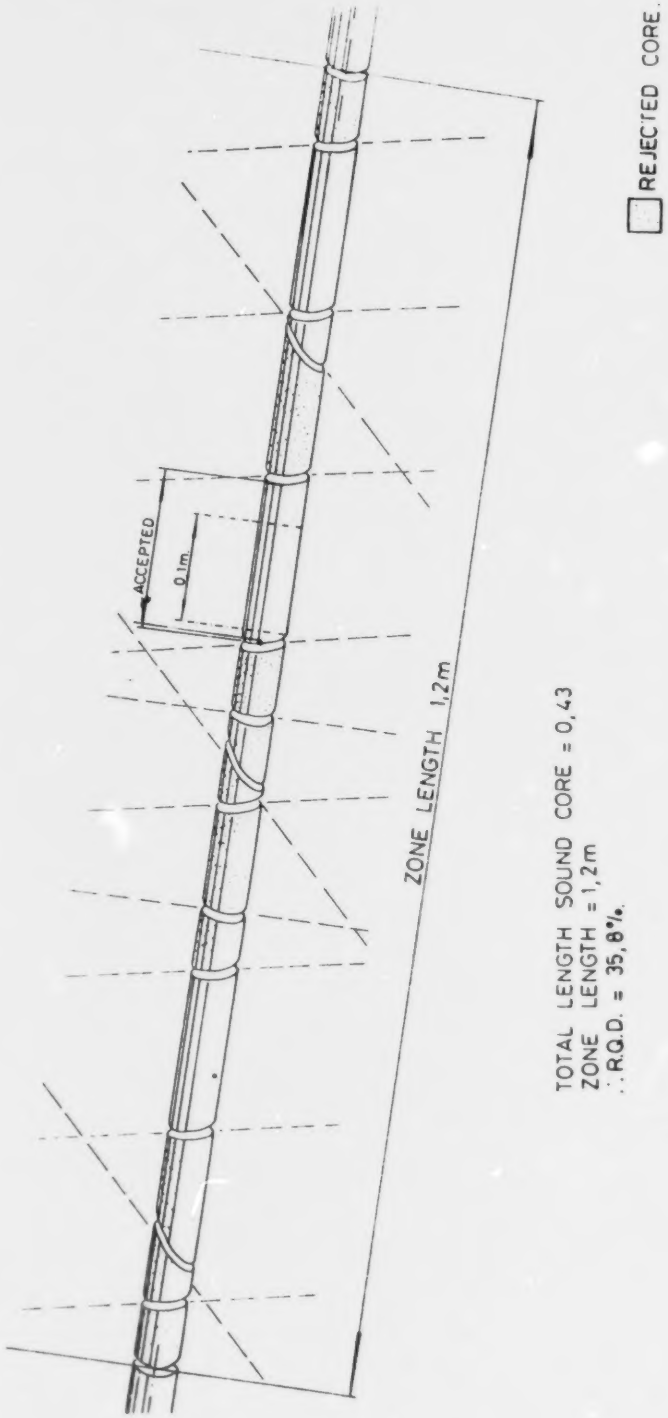
## THE METHOD OF DETERMINING ROCK QUALITY DESIGNATION (R.Q.D.%)

Figure **3.10**


REJECTED CORE.
ACCEPTED
0.1m.
ZONE LENGTH 1,2m

TOTAL LENGTH SOUND CORE = 0,43
ZONE LENGTH = 1,2m
∴ R.Q.D. = 35,8%.

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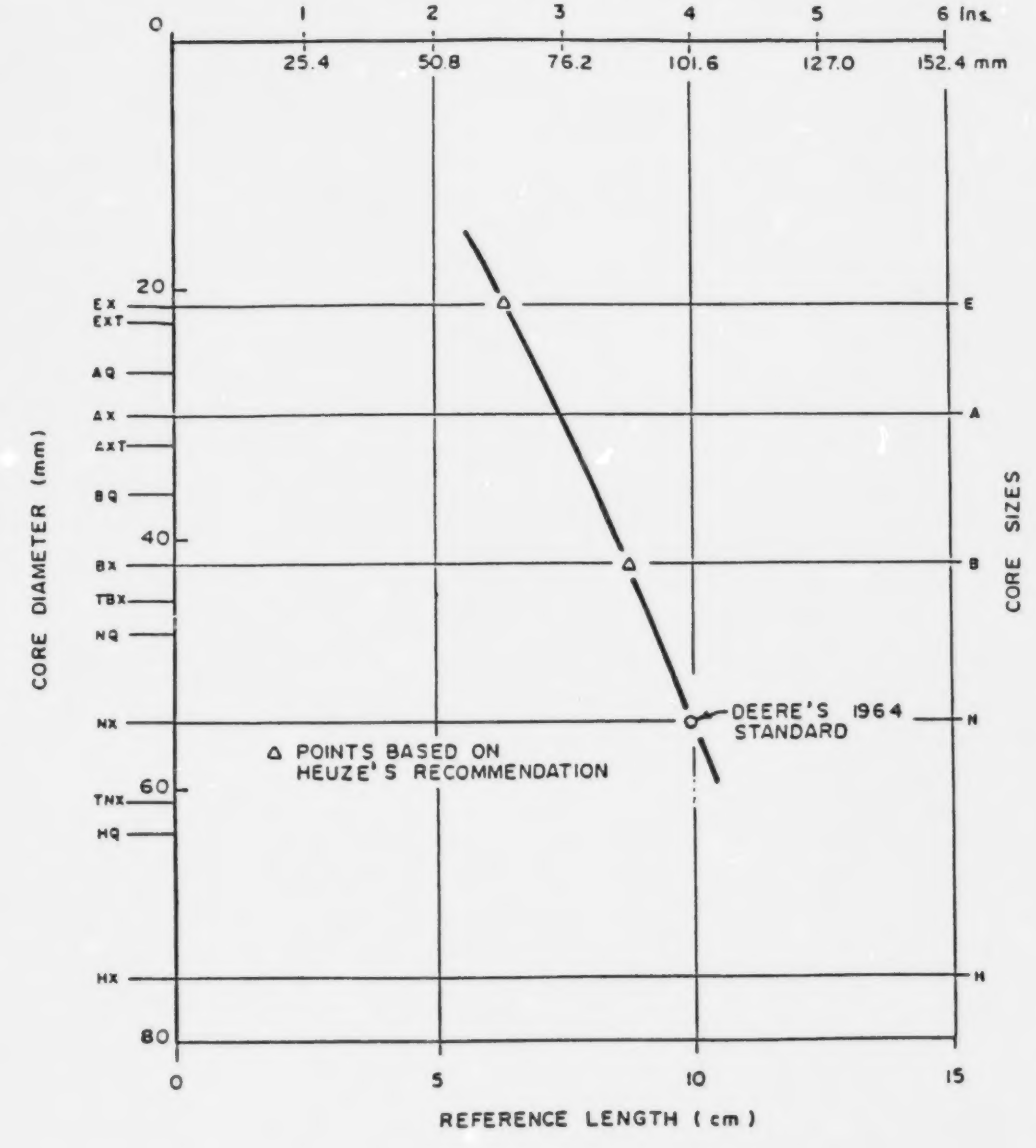
## R.Q.D. VARIATION WITH CORE DIAMETER

Figure **3.11**

RECOMMENDED CLASSIFICATION CORE LENGTH, FOR R.Q.D. DETERMINATION ( AFTER HEUZE, 1971)



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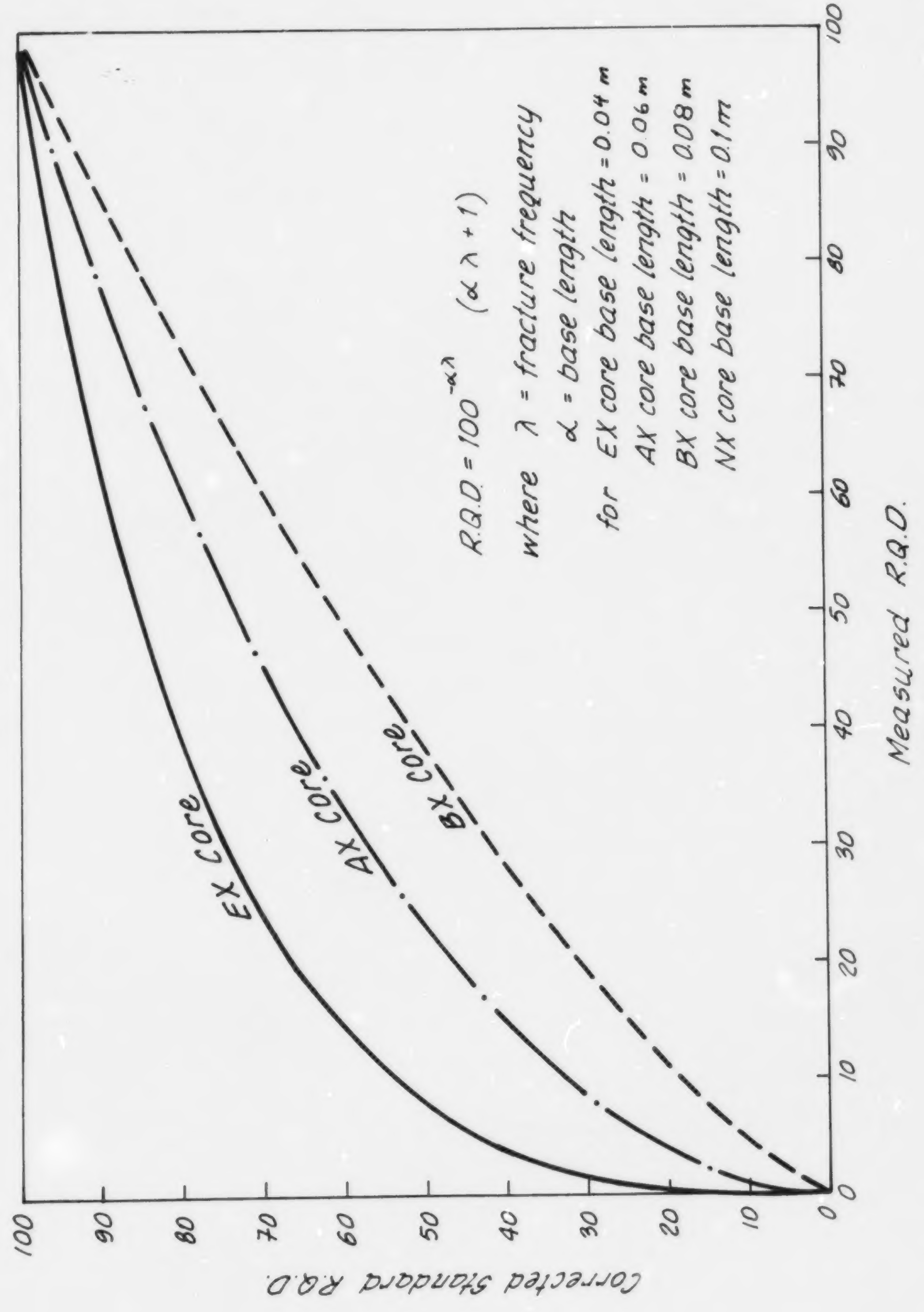
## MEASURED vs STANDARD R.Q.D. FOR DIFFERENT CORE SIZES

Figure **3.12**



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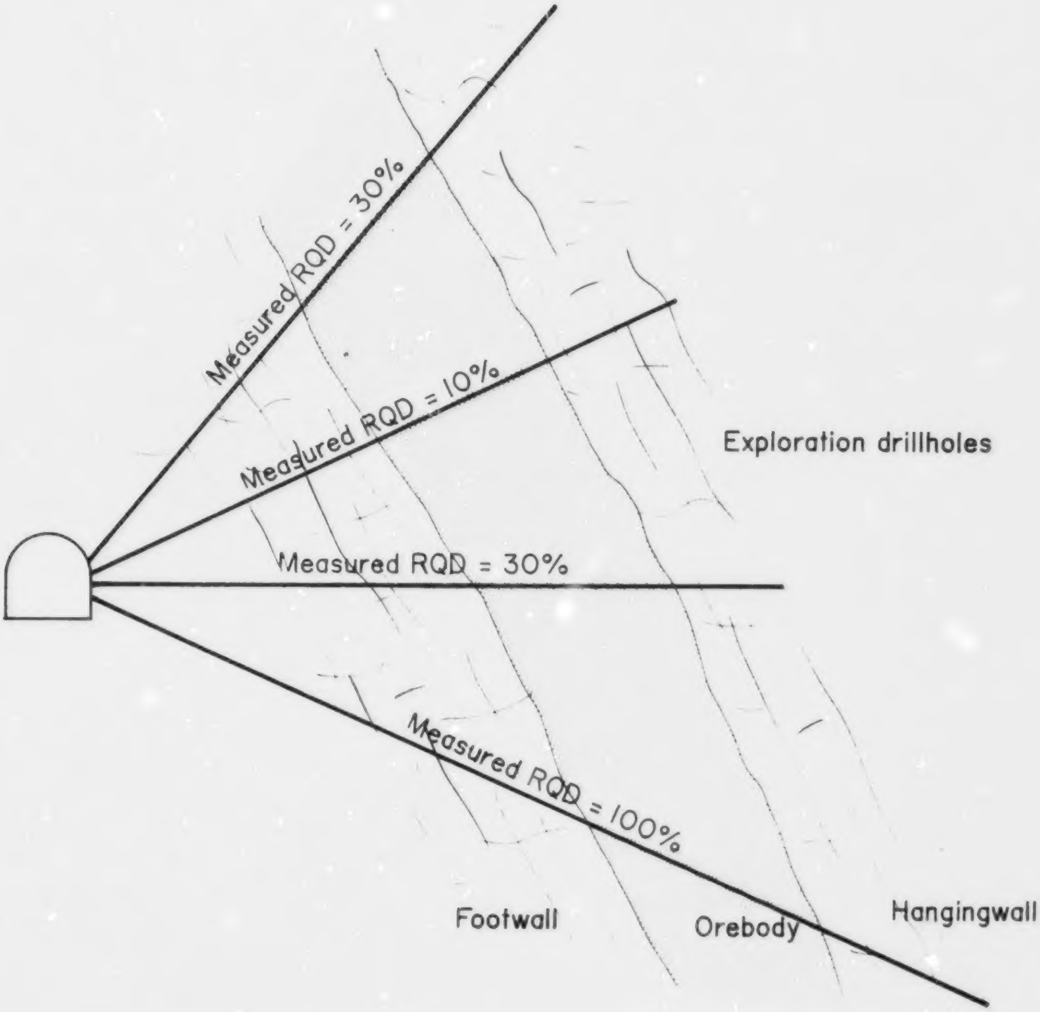
## DIRECTIONAL BIAS~RQD

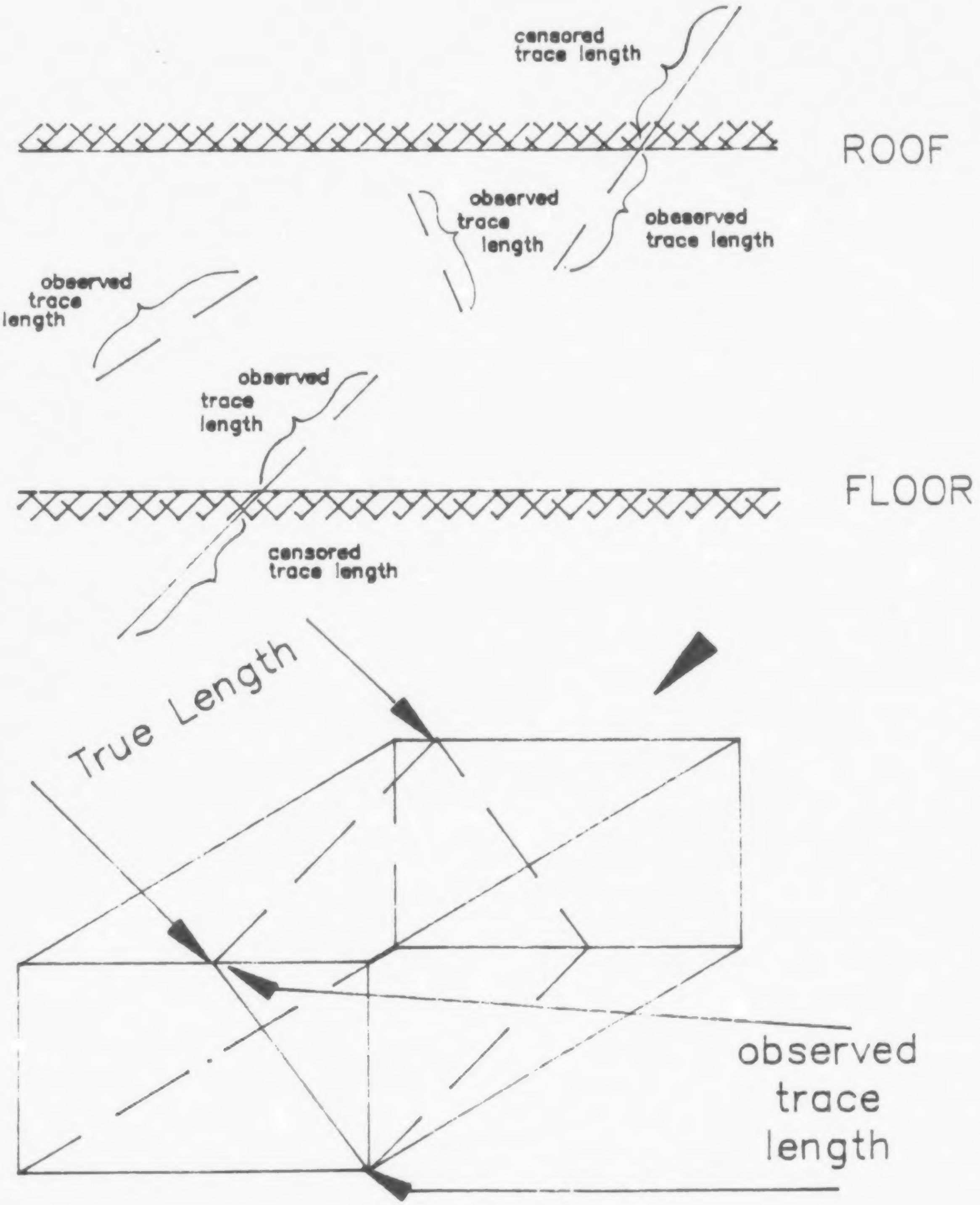
Figure **3.13**


Measured RQD = 30%
Measured RQD = 10%
Exploration drillholes
Measured RQD = 30%
Measured RQD = 100%
Footwall
Orebody
Hangingwall

SCHEMATIC ONLY
Not to Scale

PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

**CENSORSHIP OF DISCOUTINUITY TRACE LENGTH** Figure 3.14


censored trace length
ROOF
observed trace length
obeserved trace length
observed trace length
observed trace length
FLOOR
censored trace length
True Length
observed trace length

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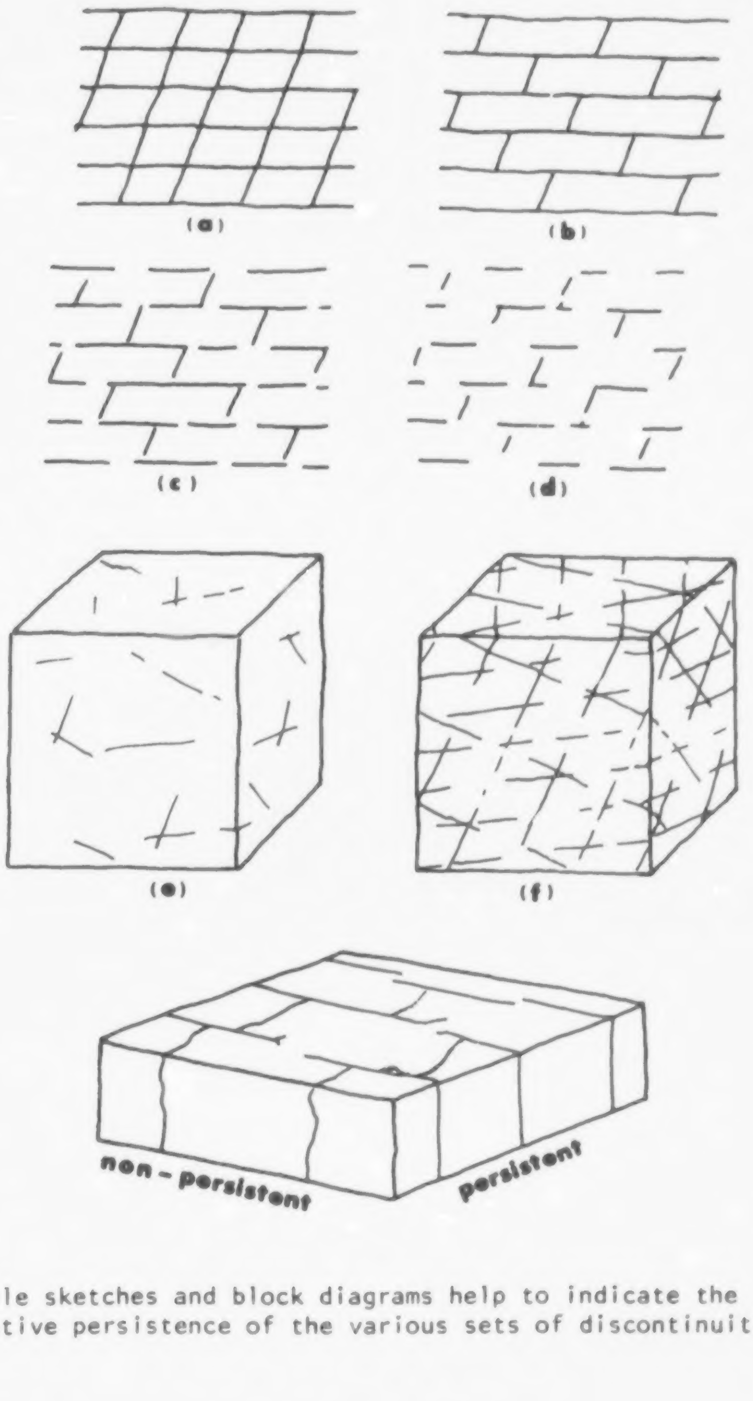
# RELATIVE PERSISTENCE

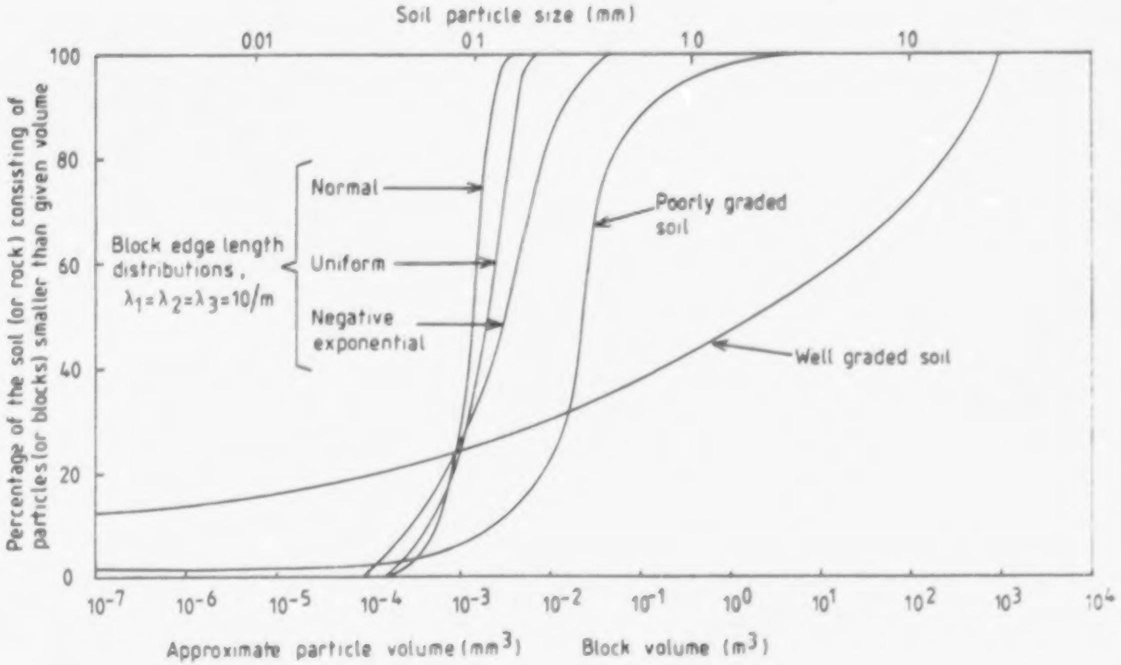
Figure **3.15**


(a)
(b)
(c)
(d)
(e)
(f)
non - persistent
persistent

Simple sketches and block diagrams help to indicate the relative persistence of the various sets of discontinuities.

# GRADING CURVES FOR SOIL PARTICLES AND ROCK BLOCKS

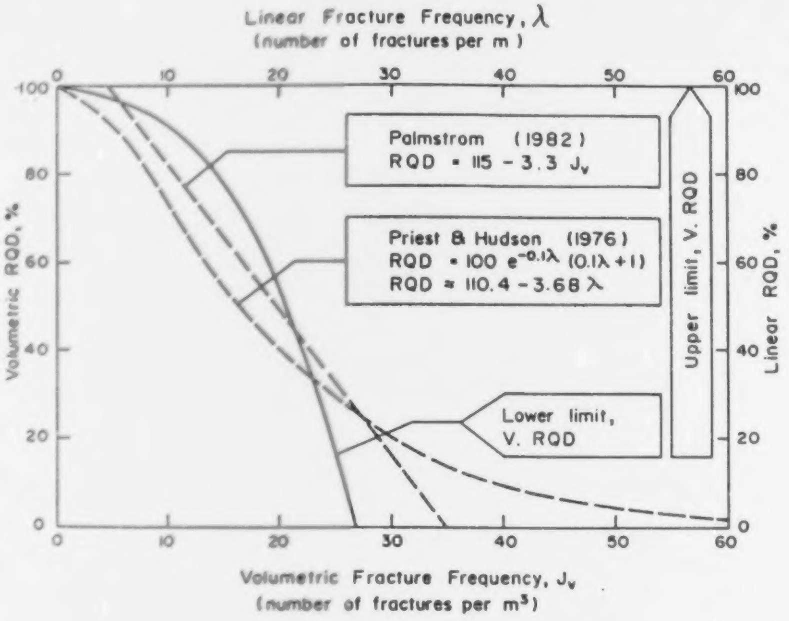
Figure **3.16**



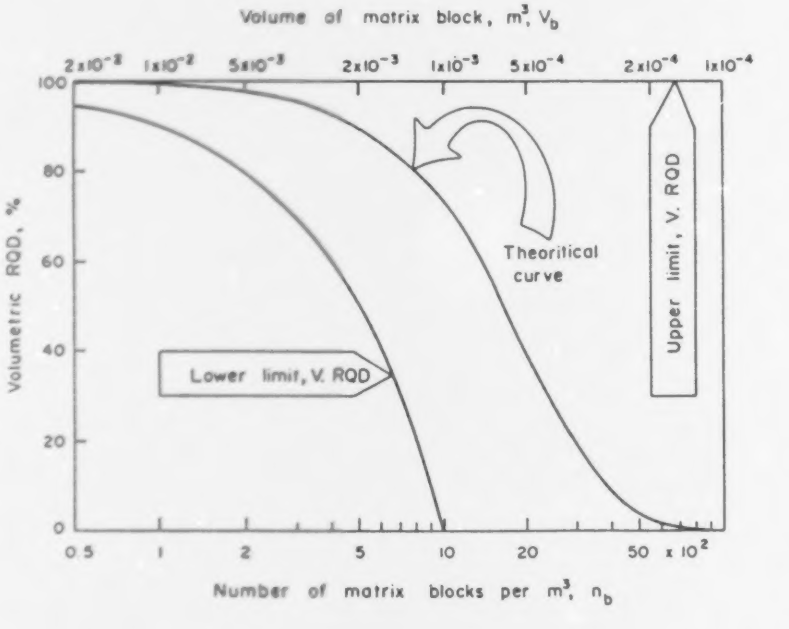
(after Hudson and Priest, 1979)

# RELATIONSHIPS BETWEEN FRACTURE FREQUENCY, VOLUMETRIC RQD and BLOCK SIZE

Figure **3.17**



Relationship between RQD and fracture frequency showing the lower and upper limits for the volumetric RQD.



Relationship between V.RQD and representative size of a matrix block.

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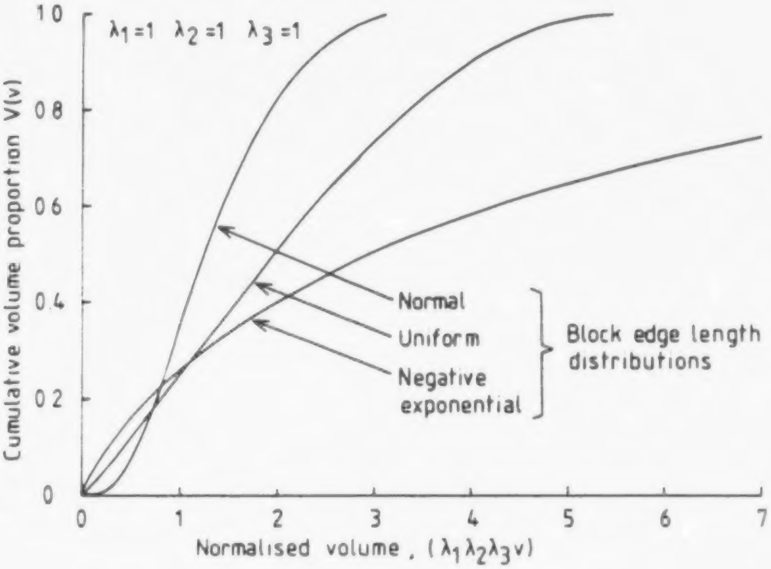
# CUMULATIVE BLOCK VOLUME PROPORTION CURVES FOR THREE ORTHOGONAL DISCONTINUITY SETS

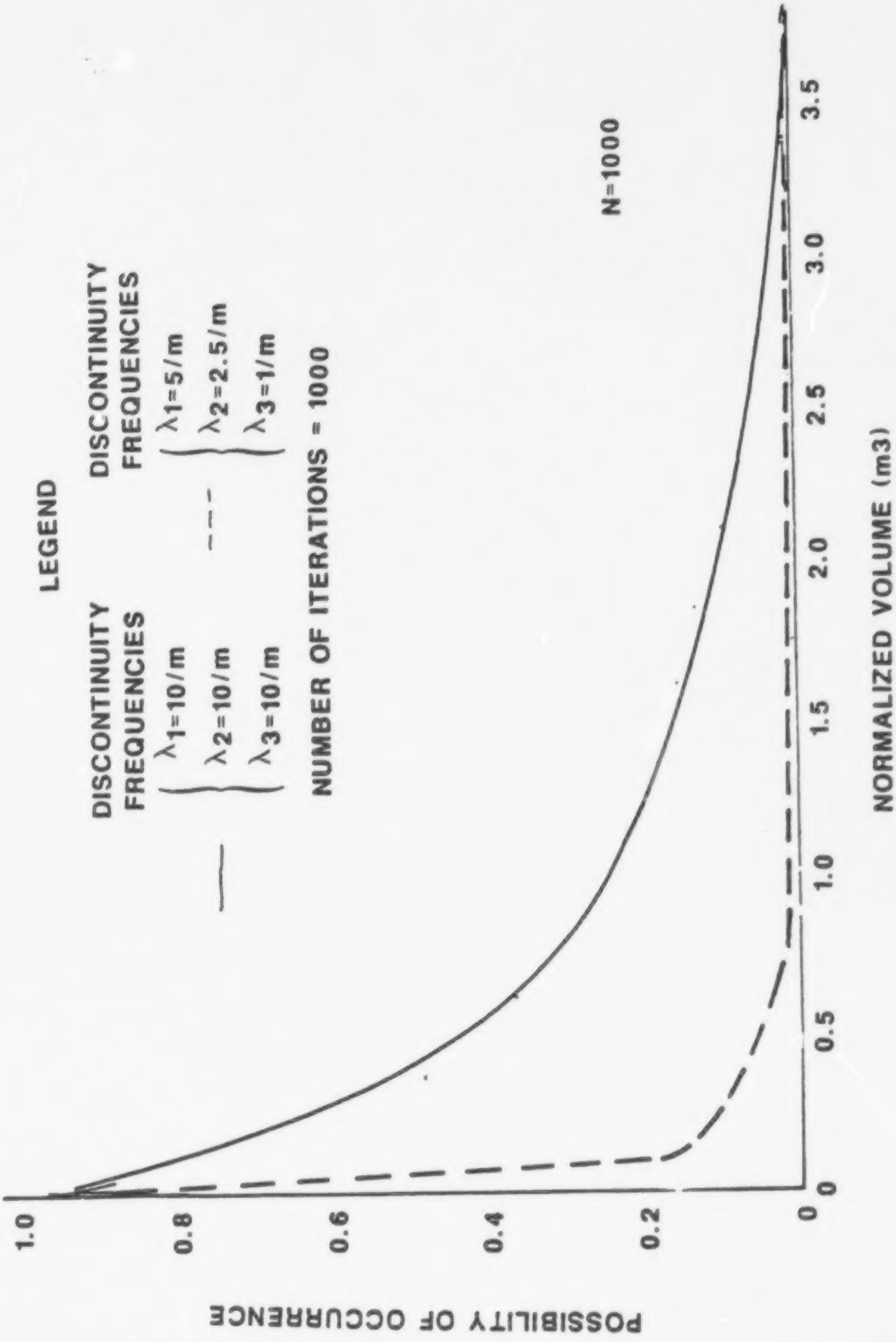
Figure **3.18**



PROJECT NO. ............ DRAWN ............ REVIEWED ............ DATE ............

## DISTRIBUTION OF BLOCK VOLUME

**Figure 3.19**



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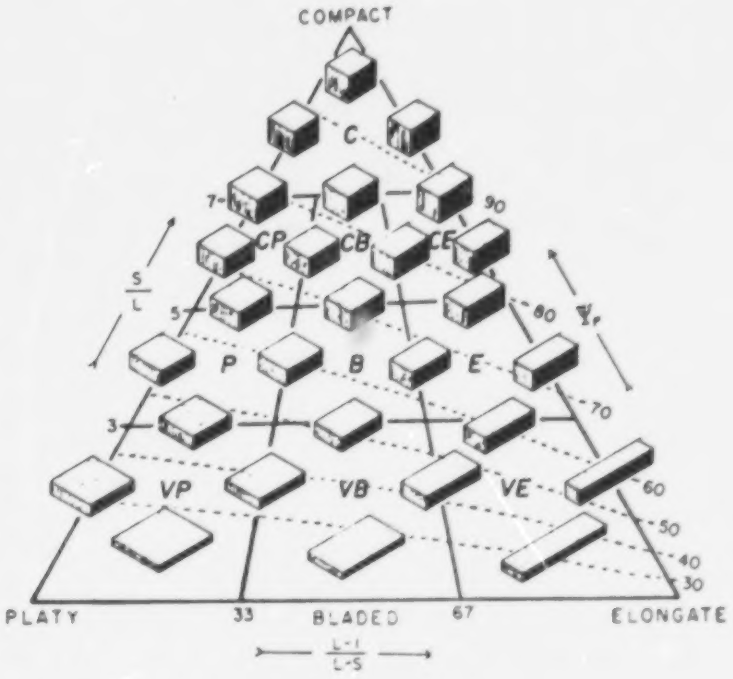
# PROPOSED CLASSIFICATION OF BLOCK SHAPE

Figure **3.20**


COMPACT
C
CP
CB
CE
P
B
E
VP
VB
VE
7
5
3
S/L
90
80
70
60
50
40
30
$\Psi_p$
PLATY
33
BLADED
67
ELONGATE
$\frac{L-I}{L-S}$

(after Folk, 1965)

C = Compact (cube)
CP = Compact Platy
P = Platy
VP = Very Platy

VB = Very Bladed
B = Bladed
CB = Compact Bladed
CE = Compact Elongate

E = Elongate
VE = Very Elongate

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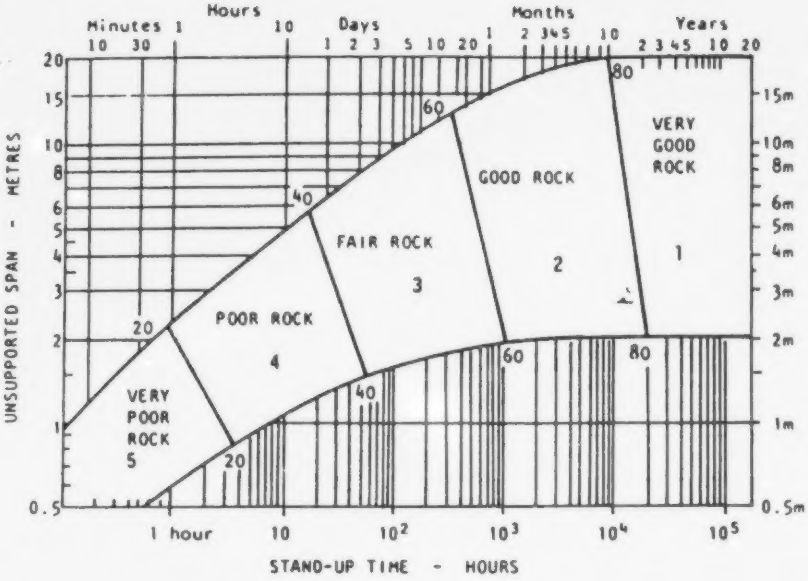
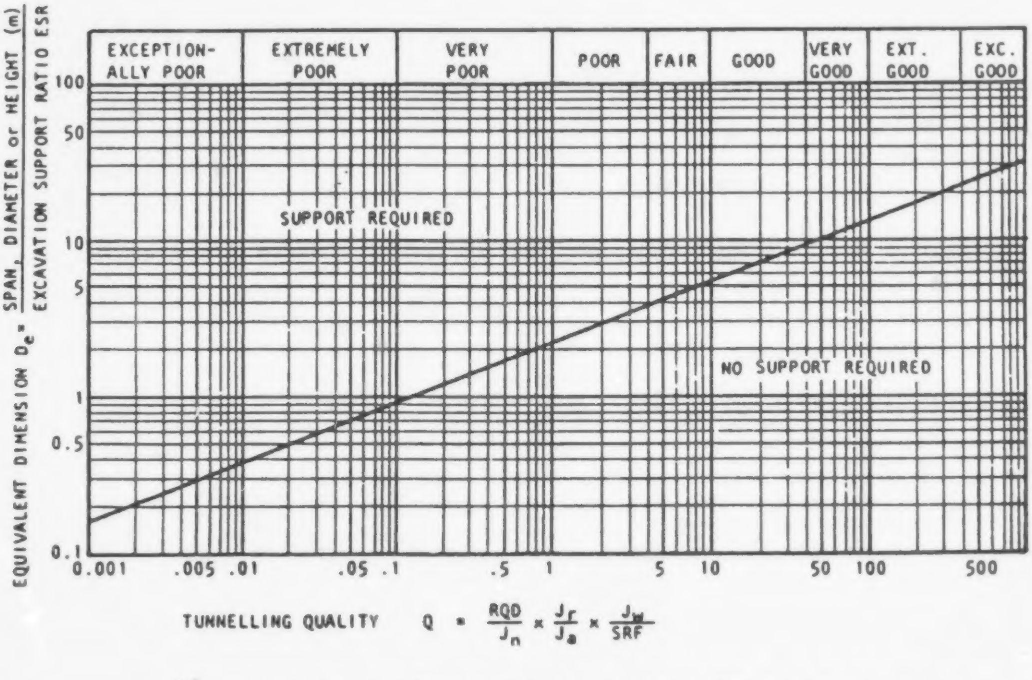
# ROCK MASS RATING AND SUPPORT CLASSIFICATION

Figure **3.21**



a) Relationship between the stand-up time of an unsupported underground excavation span and the CSIR Geomechanics Classification proposed by Bieniawski



b) Relationship between the maximum equivalent dimension $D_e$ of an unsupported underground excavation and the NGI tunnelling quality index Q. (After Barton, Lien and Lunde )

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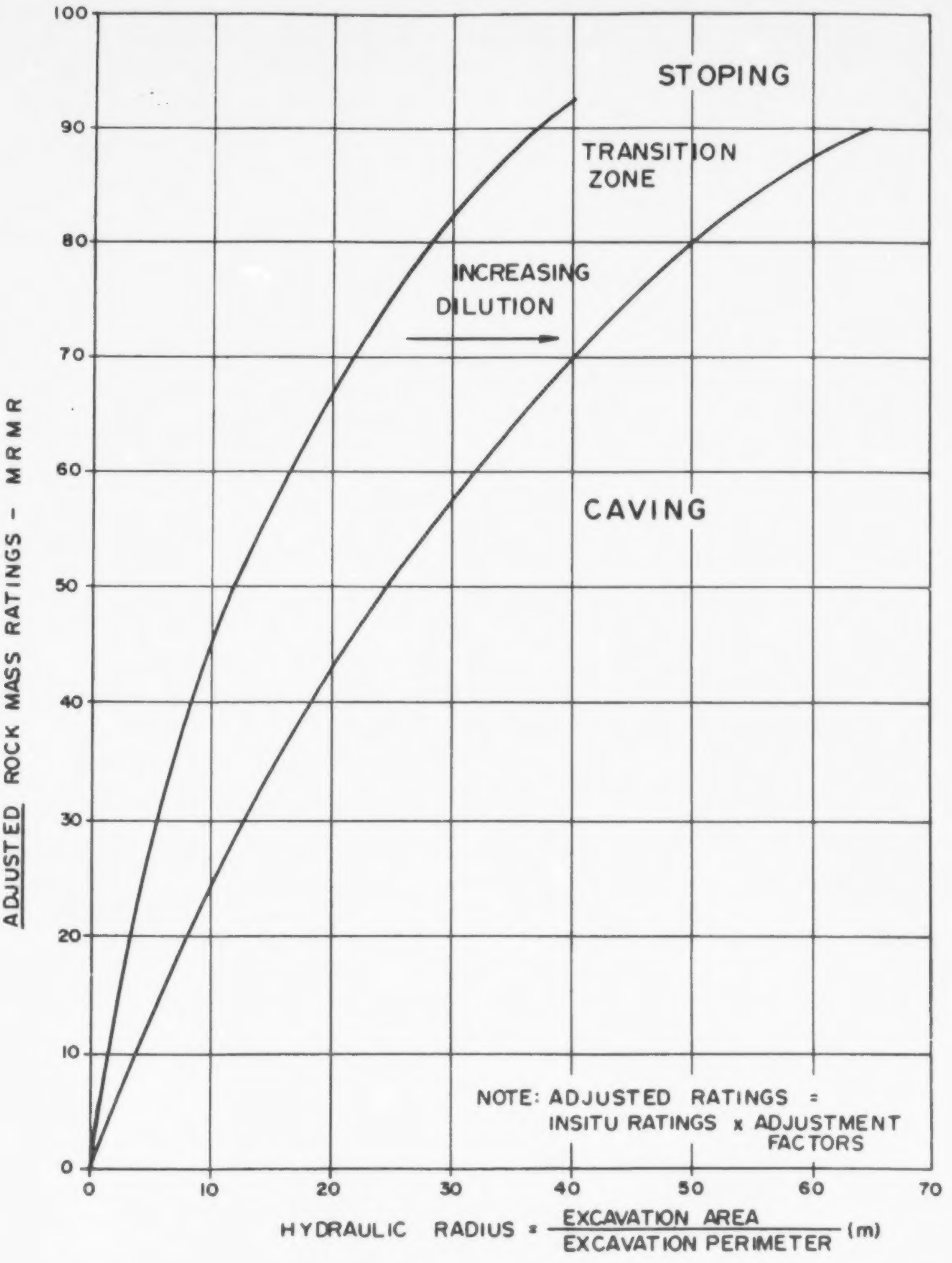
## GRAPH OF MRMR versus HYDRAULIC RADIUS

Figure **3.22**



(after Laubscher, 1987)

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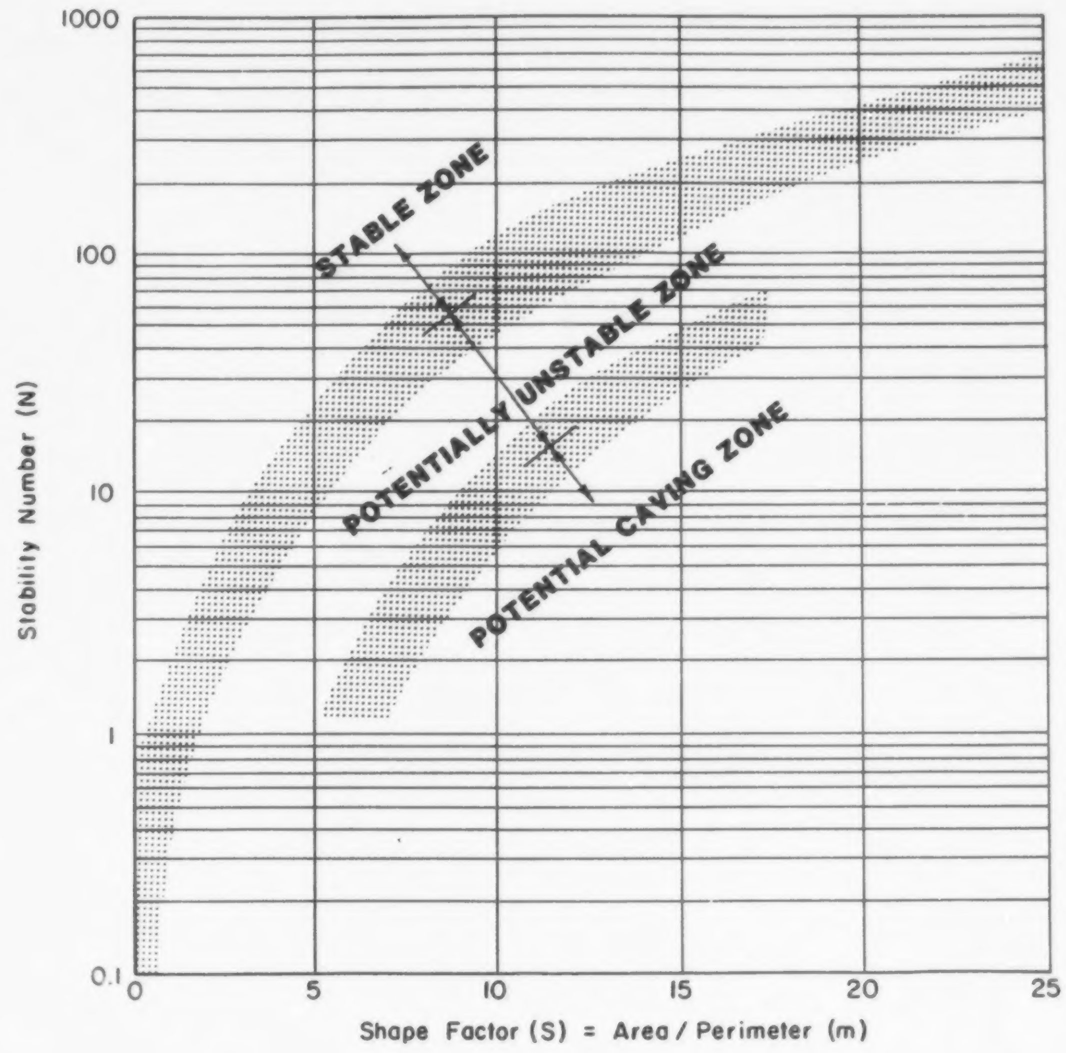
## GRAPH OF STABILITY NUMBER (N) VERSUS SHAPE FACTOR (S)

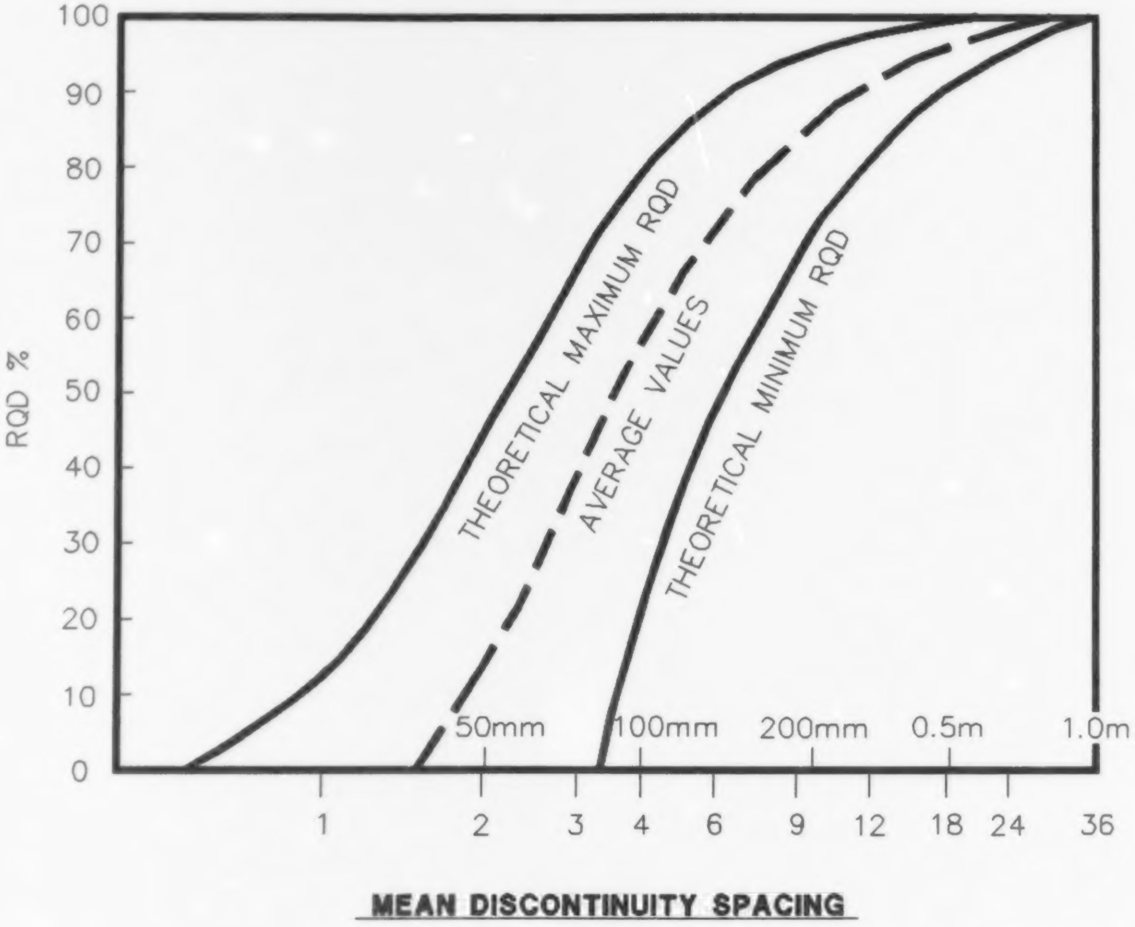
Figure 3.23



(after Mathews et al 1981)

## RQD VERSUS MEAN DISCONTINUITY SPACING

**Figure 3.24**



(after Priest and Hudson, 1976)

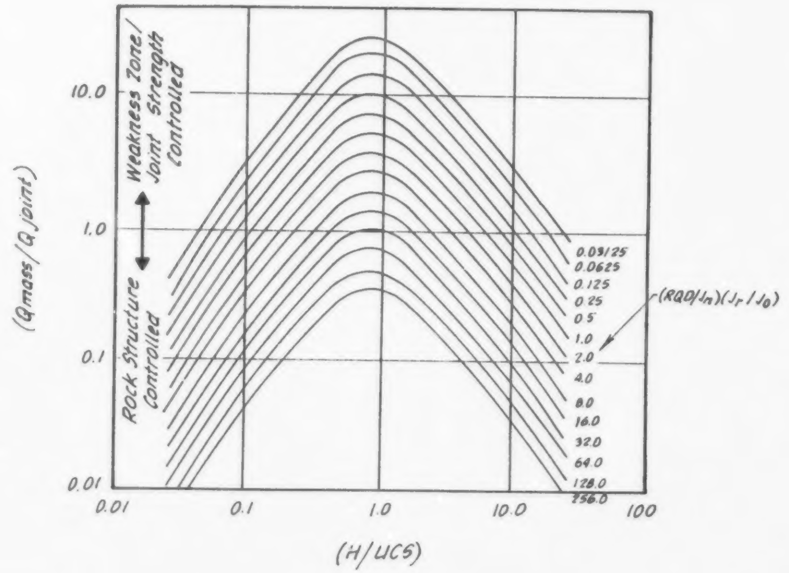
# STRESS REDUCTION FACTOR (SRF)

Figure **3.25**

| Material | Description of Intact Ground | SRF for Rock Mass Predominated, Fracturing, Squeezing or Swelling | | | | | | | | | | | | | | | | | Single Weakness or Joint Infill Predominated Confinement or Squeezing | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Unconfined Compressive Strength (MPa) | Head of Rock, H, (m) | | | | | | | | | | | | | | | | Q | SRF |
| | | | 2 | 4 | 6 | 8 | 10 | 20 | 40 | 60 | 80 | 100 | 200 | 300 | 400 | 600 | 800 | 1000 | 0.001 | 17.5 |
| Granular/ Cohesive Soil | Very loose/Very Soft | 0.035 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 0.002 | 14.0 |
| | Loose/Soft | 0.069 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 0.003 | 10.3 |
| | Medium Dense/Firm | 0.14 | 11.4 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 0.01 | 8.2 |
| | Coarse/Stiff | 0.28 | 4.6 | 11.4 | 19.5 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 0.04 | 5.2 |
| | Very Dense/Very Stiff | 0.55 | 1.9 | 4.7 | 8.0 | 11.7 | 15.7 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 0.1 | 3.9 |
| Rock | Very Soft Rock | 1.1 | 0.8 | 1.9 | 3.2 | 4.7 | 6.3 | 15.7 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 0.4 | 2.4 |
| | Medium Soft Rock | 2.2 | 0.5 | 0.8 | 1.3 | 1.9 | 2.3 | 6.3 | 15.7 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 1 | 1.8 |
| | Soft Rock | 4.4 | 0.7 | 0.5 | 0.6 | 0.8 | 1.1 | 2.3 | 6.3 | 10.7 | 15.7 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 2 | 1.4 |
| | Medium Hard Rock | 8.8 | 1.5 | 0.7 | 0.5 | 0.5 | 0.5 | 1.1 | 2.5 | 4.3 | 6.3 | 8.4 | 20 | 20 | 20 | 20 | 20 | 20 | 3 | 1.3 |
| | Hard Rock | 17.7 | 3.9 | 1.5 | 0.9 | 0.7 | 0.6 | 0.5 | 1.1 | 1.7 | 2.9 | 3.4 | 8.4 | 14.8 | 20 | 20 | 20 | 20 | 4 | 1.1 |
| | Very Hard Rock | 35.4 | 9 | 3.9 | 2.2 | 1.9 | 1.1 | 0.6 | 0.9 | 0.8 | 1.1 | 1.4 | 3.4 | 5.7 | 8.4 | 14.3 | 20 | 20 | 6 | 1.0 |
| | Very Very Hard Rock | 70.7 | 5 | 5 | 5 | 3.8 | 2.8 | 1.1 | 0.6 | 0.3 | 0.5 | 0.6 | 1.4 | 2.3 | 3.4 | 5.7 | 8.4 | 11.3 | 10 | 0.9 |
| | Extremely Hard Rock | 141.3 | 5 | 5 | 5 | 5 | 5 | 2.8 | 1.1 | 0.7 | 0.6 | 0.5 | 0.6 | 1.0 | 1.4 | 2.3 | 3.4 | 4.5 | 20 | 0.7 |
| | Exceptionally Hard Rock | 282.4 | 9 | 5 | 5 | 5 | 5 | 5 | 2.8 | 1.6 | 1.1 | 0.9 | 0.5 | 0.5 | 0.6 | 1.0 | 1.4 | 1.8 | 30 | 0.6 |

NOTES:

1. Take maximum SRF from axis or joint controlled criteria.
2. Figures given apply for $\sigma_1/\sigma_3$ less than 5.
3. For $\sigma_1/\sigma_3$ between 5 and 10 multiply the figures above the minimum diagonal by 1.9.
4. For $\sigma_1/\sigma_3$ larger than 10 multiply the figures above the minimum diagonal by 2.0.
5. For joint controlled criterion determine SRF by iterative trial and error.



Note: The SRF parameters can be estimated from Table above or inferred from graph

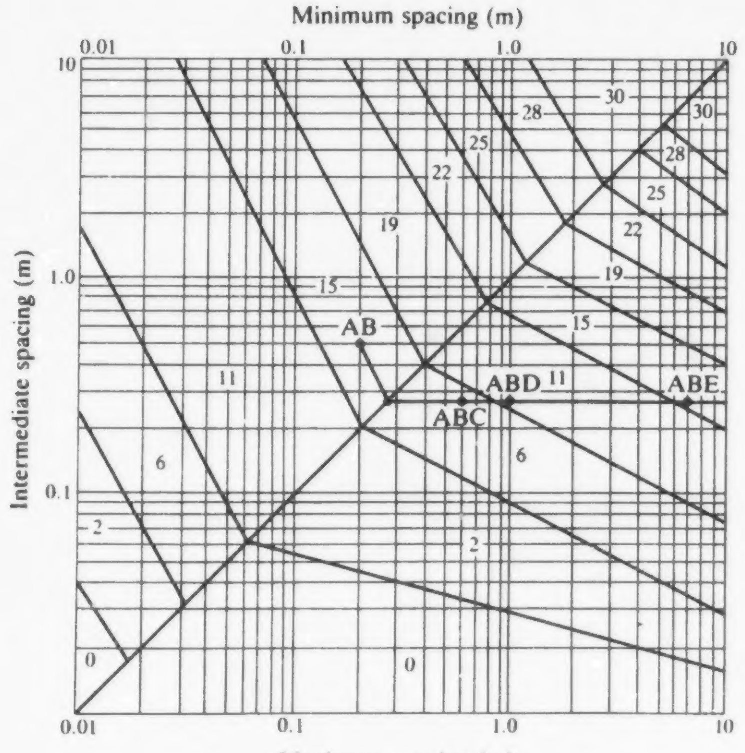
## MINING ROCK MASS CLASSIFICATION

Figure **3.26**

| | Parameter | Range of Values | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | RQD | 100—97 | | 96—84 | 83—71 | 70—56 | 55—44 | 43—31 | 30—17 | 16—4 | 3—0 | |
| | **Rating** ( = RQD × 15/100) | 15 | | 14 | 12 | 10 | 8 | 6 | 4 | 2 | 0 | |
| 2 | UCS (MPa) | 185 | 184—165 | 164—145 | 144—125 | 124—105 | 104—85 | 84—65 | 64—45 | 44—25 | 24—5 | 4—0 |
| | **Rating** | 20 | 18 | 16 | 14 | 12 | 10 | 8 | 6 | 4 | 2 | 0 |
| 3 | Joint Spacing | Refer Figure 3.27 | | | | | | | | | | |
| | **Rating** | 25 ———— 0 | | | | | | | | | | |
| 4 | Joint Condition Including Groundwater | Refer Figure 3.28 | | | | | | | | | | |
| | **Rating** | 40 ———— 0 | | | | | | | | | | |

## JOINT SPACING RATING FOR MULTI-JOINT SYSTEMS

Figure **3.27**



NOTE: In the example, spacing of sets A, B, C, D and E are 0.2, 0.5, 0.6, 1.0 and 7.0 m respectively; the combined ratings for AB, ABC, ABD and ABE are 15, 6, 11 and 15 respectively.

(after Laubscher, 1`77)

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# ADJUSTMENTS FOR JOINT CONDITION AND GROUNDWATER

Figure **3.28**

| Parameter | Description | | Dry condition | Wet Conditions | | |
|---|---|---|---|---|---|---|
| | | | | Moist | Moderate pressure 25—125 l/min | Severe pressure > 125 l/min |
| A Joint expression (large scale irregularities) | Wavy | Multi-directional | 100 | 100 | 95 | 90 |
| | | Uni-directional | 95<br>90 | 95<br>90 | 90<br>85 | 80<br>75 |
| | Curved | | 89<br>80 | 85<br>75 | 80<br>70 | 70<br>60 |
| | Straight | | 79<br>70 | 74<br>65 | 60 | 40 |
| B Joint expression (smale scale irregularities or roughness) | Very rough | | 100 | 100 | 95 | 90 |
| | Striated or rough | | 99<br>85 | 99<br>85 | 80 | 70 |
| | Smooth | | 84<br>60 | 80<br>55 | 60 | 50 |
| | Polished | | 59<br>50 | 50<br>40 | 30 | 20 |
| C Joint Wall alteration zone | Stronger than wall rock | | 100 | 100 | 100 | 100 |
| | No alteration | | 100 | 100 | 100 | 100 |
| | Weaker than wall rock | | 75 | 70 | 65 | 60 |
| D Joint filling | No fill — surface staining only | | 100 | 100 | 100 | 100 |
| | Non-softening and sheared material (clay or talc free) | Coarse sheared | 95 | 90 | 70 | 50 |
| | | Medium sheared | 90 | 85 | 65 | 45 |
| | | Fine sheared | 85 | 80 | 60 | 40 |
| | Soft sheared material (eg talc) | Coarse sheared | 70 | 65 | 40 | 20 |
| | | Medium sheared | 65 | 60 | 35 | 15 |
| | | Fine sheared | 60 | 55 | 30 | 10 |
| | Gouge thickness <amplitude of irregularity | | 40 | 30 | 10 | |
| | Gouge thickness >amplitude of irregularity | | 20 | 10 | Flowing material 5 | |

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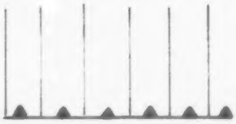
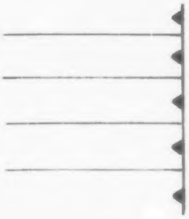
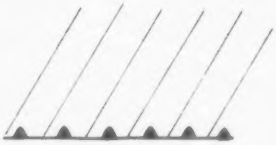
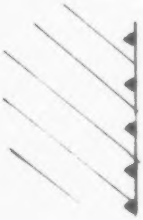
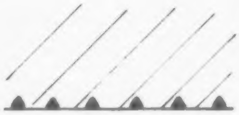
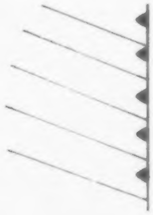
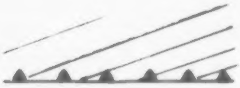
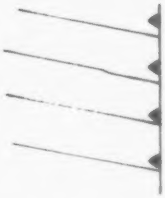
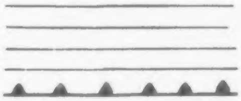
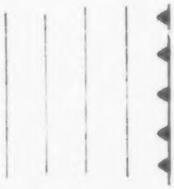
# ORIENTATION ADJUSTMENT (MODIFIED FROM LAUBSCHER, 1984)

Figure **3.29**

| Dip of Structure | Back/Roof | Adjustment | Wall | Adjustment |
|---|---|---|---|---|
| 90° |  | 100% |  | 100% |
| 60° |  | 90% |  | 70% |
| 45° |  | 75% |  | 80% |
| 20° |  | 70% |  | 90% |
| 0 |  | 80% |  | 80% |

Note: Structure refers to plunge of specific discontinuity set or the plunge of intersection between two major joints sets.

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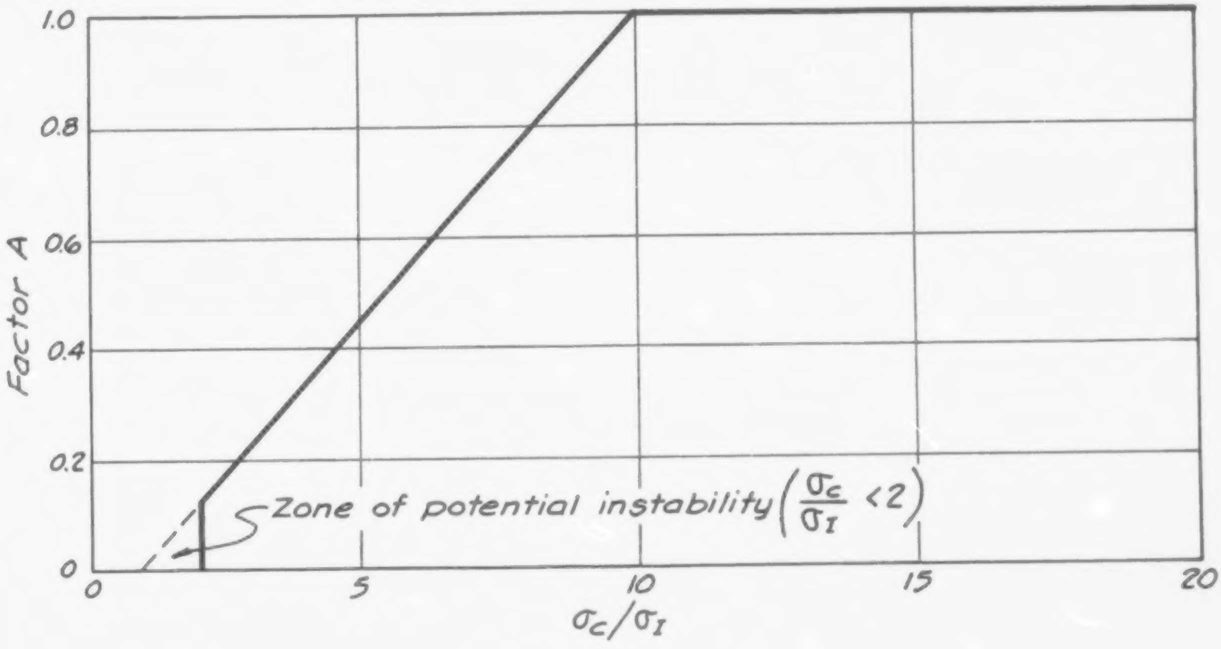
# GRAPH TO DETERMINE THE ROCK STRESS FACTOR (A)

Figure **3.30**

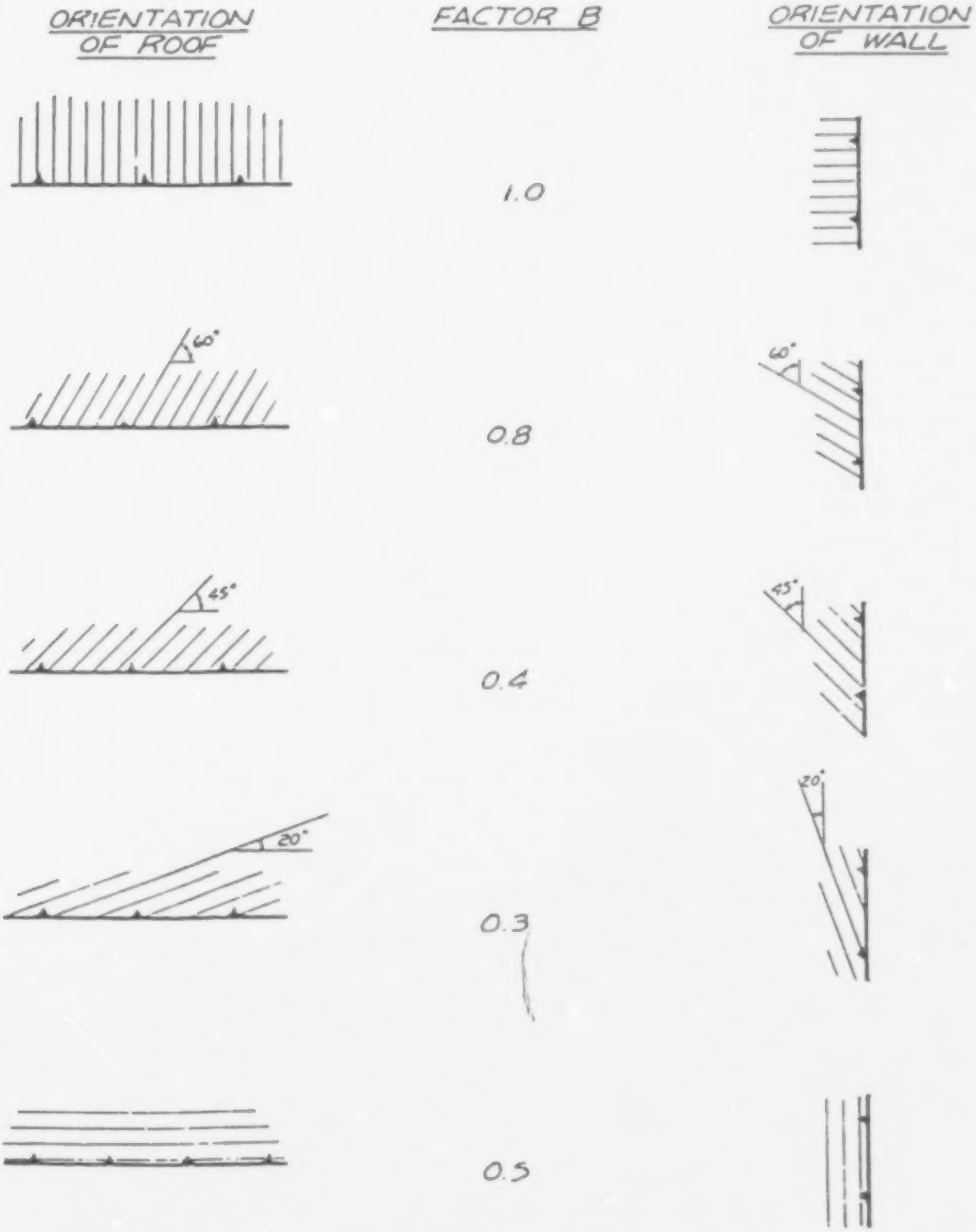


$\sigma_c$ - Uniaxial compressive strength of intact rock.
$\sigma_I$ - Induced compressive stress.

PR ........ DRAWN ........ REVIEWED ........ DATE ........

# ROCK DEFECT ORIENTATION FACTOR (B)

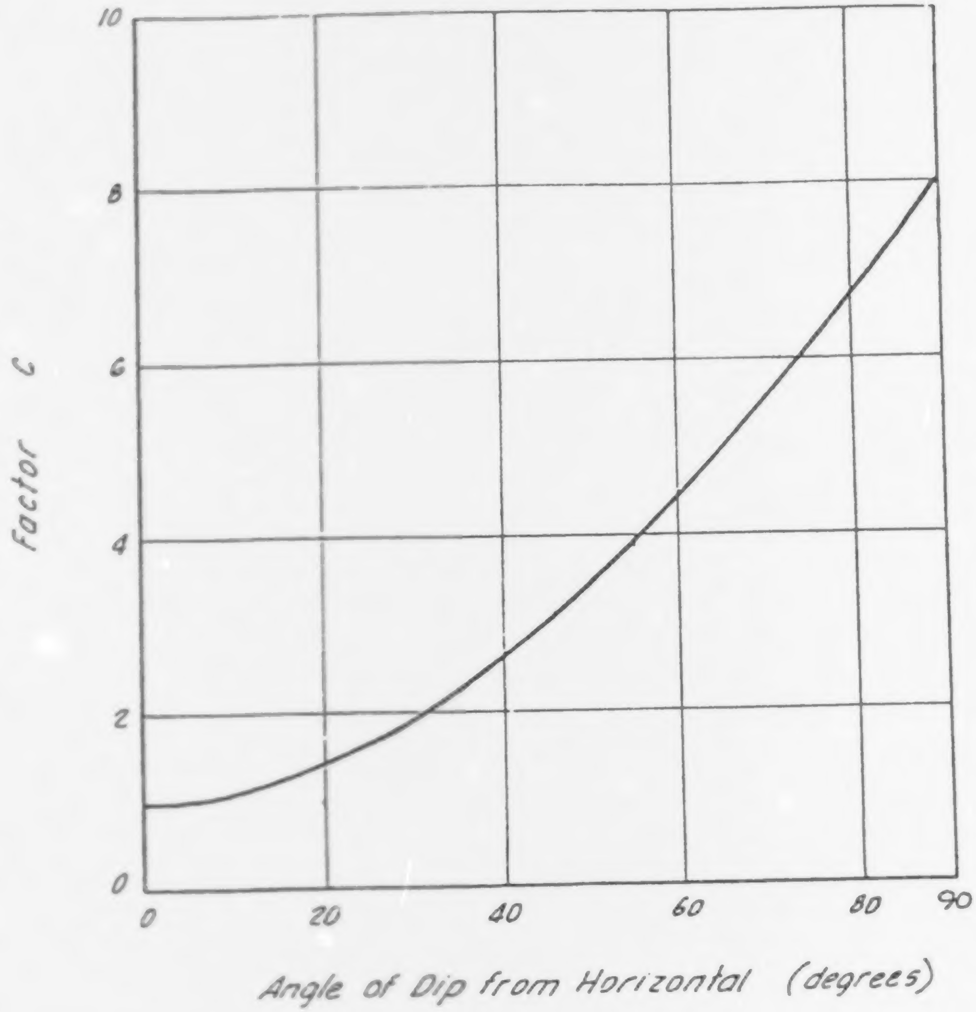
Figure **3.31**


ORIENTATION OF ROOF
FACTOR B
ORIENTATION OF WALL
1.0
60°
0.8
60°
45°
0.4
45°
20°
0.3
20°
0.5

PROJECT NO ........ DRAWN ........ REVIEWED ........ DATE ........

## DESIGN SURFACE ORIENTATION FACTOR (C)

Figure **3.32**

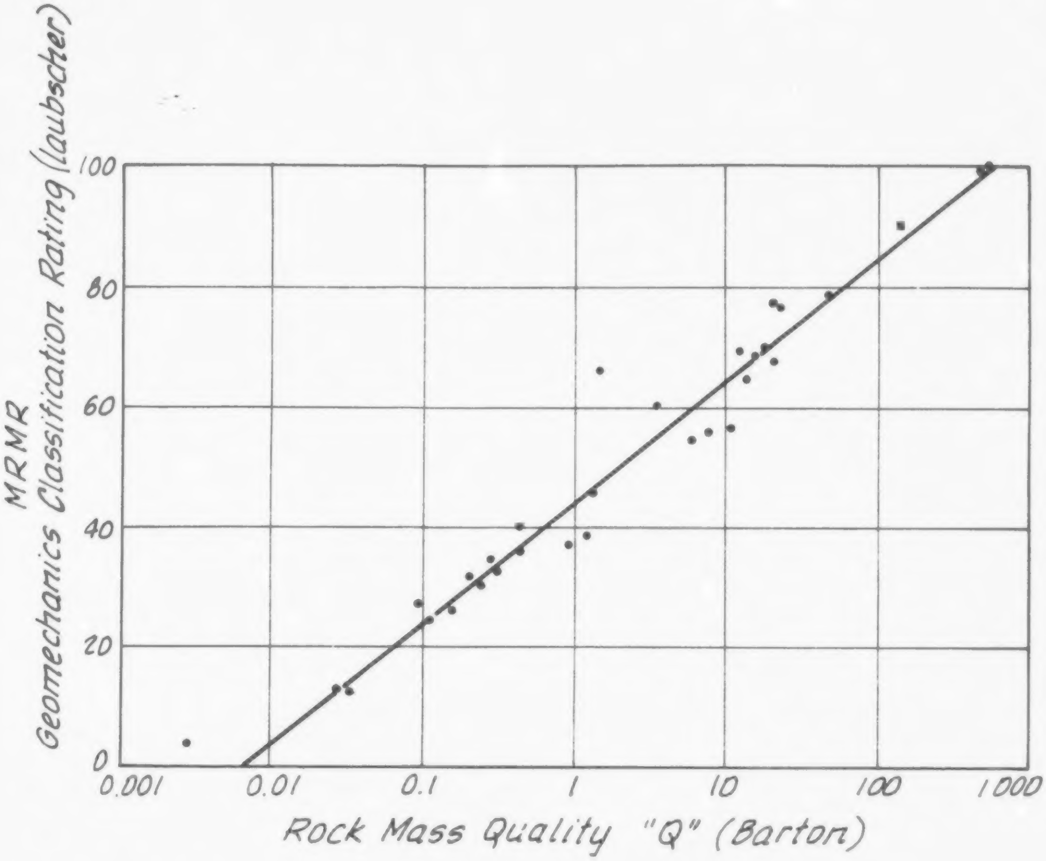


Factor C = 8 - 7 Cosine (angle of dip)

PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

# CORRELATION BETWEEN ROCK MASS CLASSIFICATION SYSTEMS

Figure **3.33**



a) Correlation between RMR (Bieniawski) and Q (Barton) classification systems.

b) Correlation between MRMR (Laubscher) and Q (Barton) classification systems.

$$RMR = 9 \log_e Q + 44$$

PROJECT NO ........ DRAWN ........ REVIEWED ........ DATE ........

# DATA COLLECTION SHEET
# ROCK MASS CLASSIFICATION

Figure 4.1

## LOCATION

Section ☐ Level ☐

Panel ☐

## PILAR GEOMETRY & CLASSIFICATION

Height ☐ Length ☐ Width ☐

Classification: Class 1 | Class 2 | Class 3 | Class 4 | Class 5

## ROCK MASS CLASSIFICATION

| Joint Data | Set 1 | Set 2 | Set 3 | Set 4 |
|---|---|---|---|---|
| Dip Direct/Dip Angle | | | | |
| Average Spacing (ft) | | | | |

Estimated No. of Joints/m$^{3}$ =

## JOINT ALTERNATION NUMBER (Ja)

| Description of gouge | Joint alteration number (Ja) for joint separation (mm) | | |
|---|---|---|---|
| | <1.01$^{1}$ | 1.0-5.0$^{2}$ | >5.0$^{3}$ |
| Tightly healed, hard, nonsoftening impermeable filling | 0.75 | - | - |
| Unaltered joint walls, surface staining only | 1.0 | - | - |
| Slightly altered, non-softening, non-cohesive rock mineral or crushed rock filling | 1.0 | 4.0 | 6.0 |
| Non-softening, slightly clayey non-cohesive filling | 3.0 | 2.0 | 10.0 |
| Softening or low friction clay mineral coatings and small quantities of swelling clays | 4.0 | 8.0$^{5}$ | 13.0 |
| Shattered or micro-shattered (swelling) clay gouge, with or without crushed rock | 5.0 | 10.0$^{4}$ | 18.0 |

## JOINT ROUGHNESS NUMBER (Jr)

| Joint Separation | Condition of Joint | Joint Roughness Number (Jr) |
|---|---|---|
| Joints/fissures tight or losing during excavation | Discontinuous joint/fissures | 1.0 |
| | Rough or irregular, indulating | 3.0 |
| | Smooth indulating | 2.0 |
| | Rough or irregular, planar | 1.5 |
| | Smooth planar | 1.0 |
| | Slickensided planar | 0.5 |
| Joints/fissures open and remain open during excavation | Joints/fissures either open or containing relatively soft gouge or sufficient thickness to prevent joint/fissure wall contact upon excavation. | 1.0 |
| | Shattered or micro-shattered clays | 1.0 |

## JOINT SET NUMBER (Jn)

| Number of joint sets | Joint set number (Jn) |
|---|---|
| Intact, no or few joint/fissures | 0.5 - 1.0 |
| One joint/fissure set | 2 |
| One joint/fissure set plus random | 3 |
| Two joint/fissure sets | 4 |
| Two joint/fissure sets plus random | 6 |
| Three joint/fissure sets | 9 |
| Three joint/fissure sets plus random | 12 |
| Four or more joint/fissure sets, random, heavily jointed, sugar cube, etc. | 15 |
| Multiple joint/fissure sets (crushed rock, earth-like) | 20 |

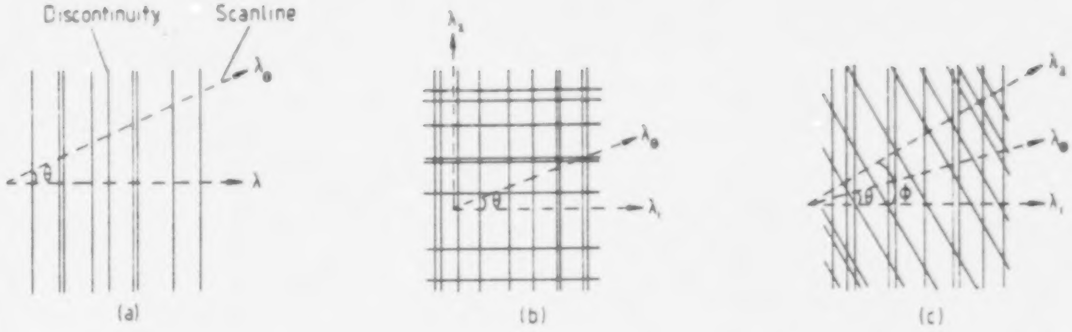
## STRESS REDUCTION FACTOR (SRF)

| | | |
|---|---|---|
| o | Weakness zones intersecting pillar. | 10.0 |
| o | Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock (any depth). | 2.5 |
| o | Single weakness zones containing clay, or chemically disintegrated rock. | 2.5 |
| o | Multiple shear zones in competent rock (clay free), loose surrounding rock. | 7.5 |
| o | Single shear zones in competent rock (clay free). | 2.5 |
| o | Loose open joints, heavily jointed or 'sugar cube'. | 5.0 |

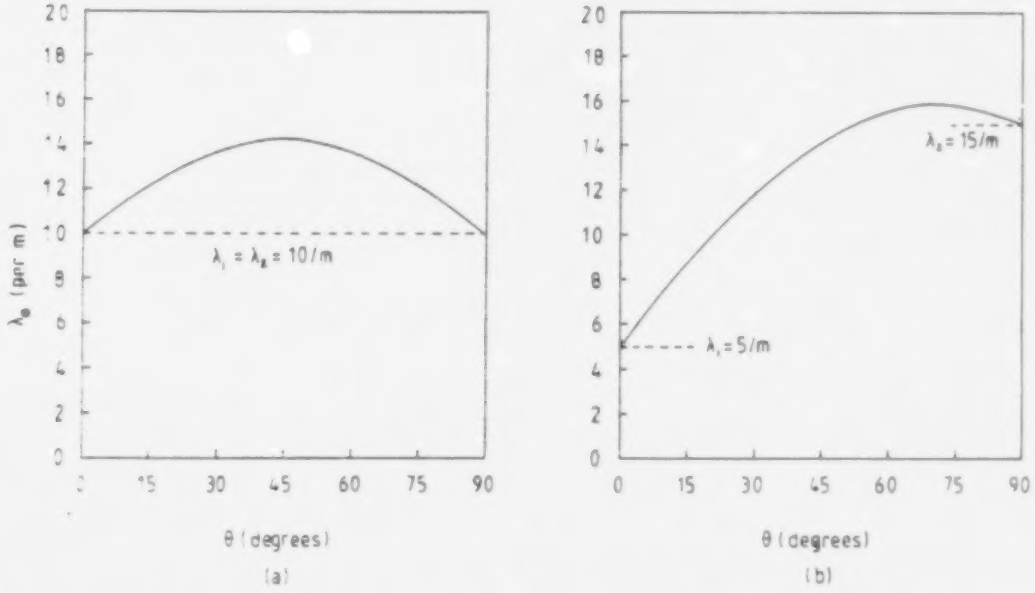
PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

# INFLUENCE OF REFERENCE LINE ORIENTATION ON FRACTURE FREQUENCY

Figure **4.2**


Discontinuity
Scanline
$\lambda_\theta$
$\theta$
$\lambda$
(a)
$\lambda_2$
$\lambda_\theta$
$\theta$
$\lambda_1$
(b)
$\lambda_2$
$\lambda_\theta$
$\theta$
$\Phi$
$\lambda_1$
(c)

Discontinuity frequency along a line in a plane intersected by discontinuity sets.



Variation of discontinuity frequency as a scanline is rotated in a plane through two orthogonal discontinuity sets.

(from Hudson and Priest, 1983)

PROJECT NO ........ DRAWN ........ REVIEWED ........ DATE ........

PROJECT NO .................... DRAWN ........ REVIEWED ........ DATE ....................

# DATA COLLECTION SHEET - CORE LOGGING

Figure **4.3**

SITE: ____________ NORTHING ____________ INCLINATION: ____________ DATE

STARTED: ____________ BOREHOLE: ________

LOCATION: ____________ EASTING ____________ BEARING: ____________ COMPLETION: ____________ PAGE ___ OF ___

ELEVATION: ____________

DRILLING METHOD: ____________ DRILLING CONTRACTOR: ____________ LOGGED BY: ____________

| | | | | | DIP OF FRACTURES | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| DEPTH | WEATH ALT | ROCK TYPE | RQD | TOTAL CORE RECOVERY | 90 - 70 | | | | 70 - 50 | | | | 50 - 30 | | | | 30 - 0 | | | | STRENGTH | | | COMMENTS |
| | | | | | NO | ROUGH | ALTER | TYPE | NO | ROUGH | ALTER | TYPE | NO | ROUGH | ALTER | TYPE | NO | ROUGH | ALTER | TYPE | | | | |
| | | | | | | | | | | | | | | | | | | | | | | | | |
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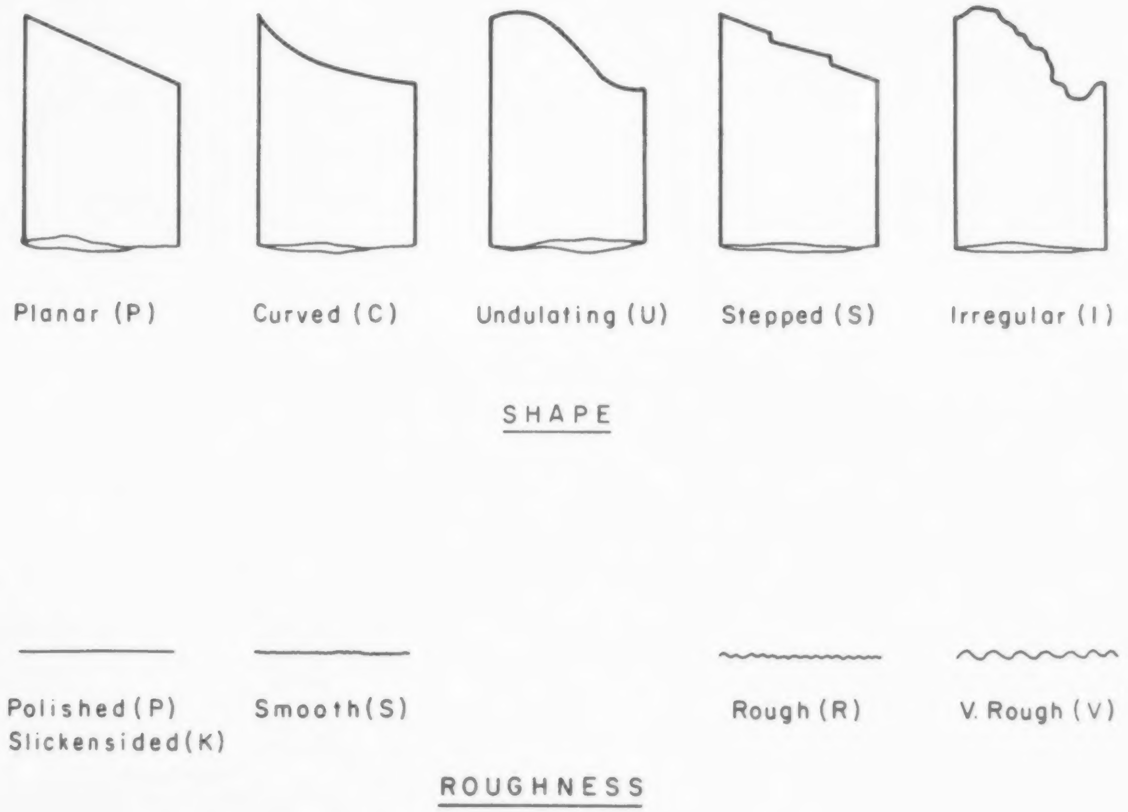
## SHAPE AND ROUGHNESS OF JOINTS IN DRILL CORE

Figure **4.4**

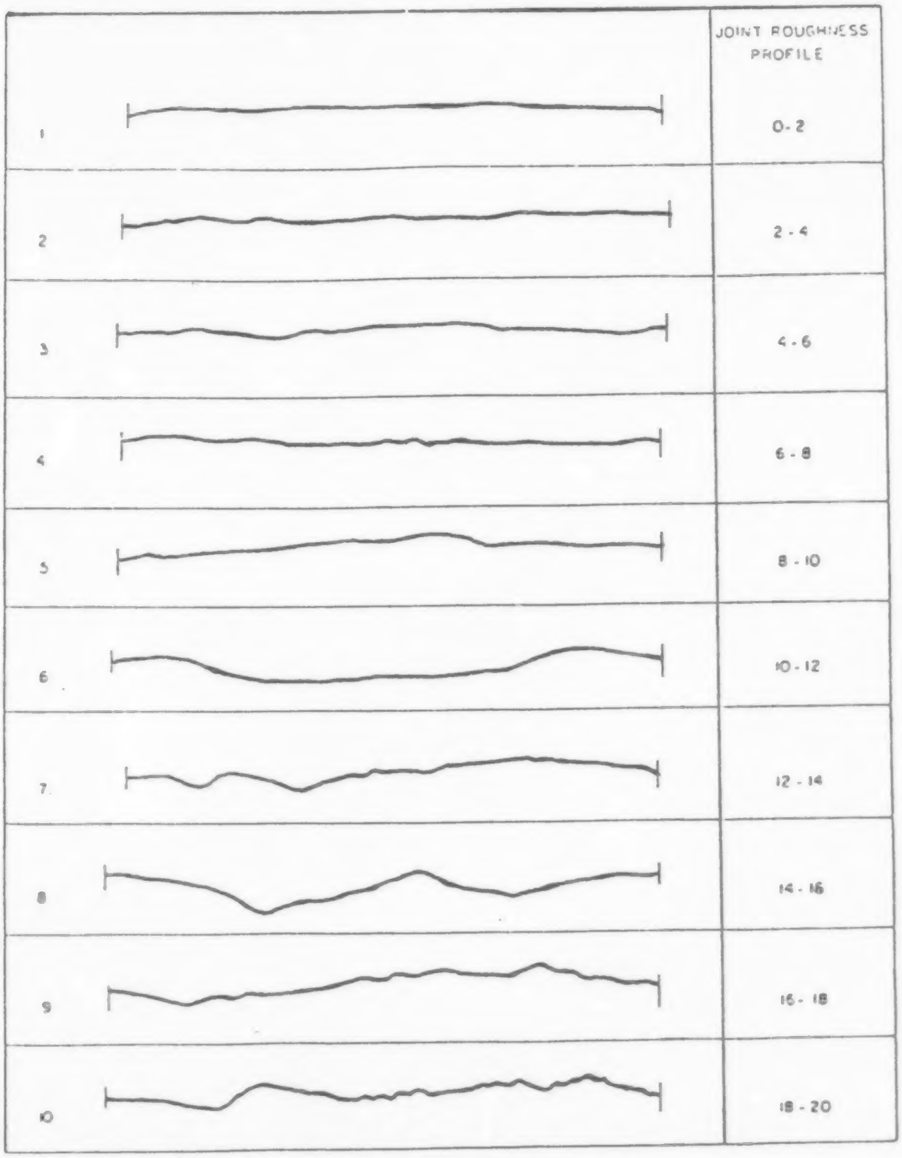


Planar (P)
Curved (C)
Undulating (U)
Stepped (S)
Irregular (I)
SHAPE
Polished (P)
Slickensided (K)
Smooth (S)
Rough (R)
V. Rough (V)
ROUGHNESS

PROJECT NO ............ DRAWN ............ REVIEWED ............ DATE ............

# ROUGHNESS PROFILES

Figure 4.5

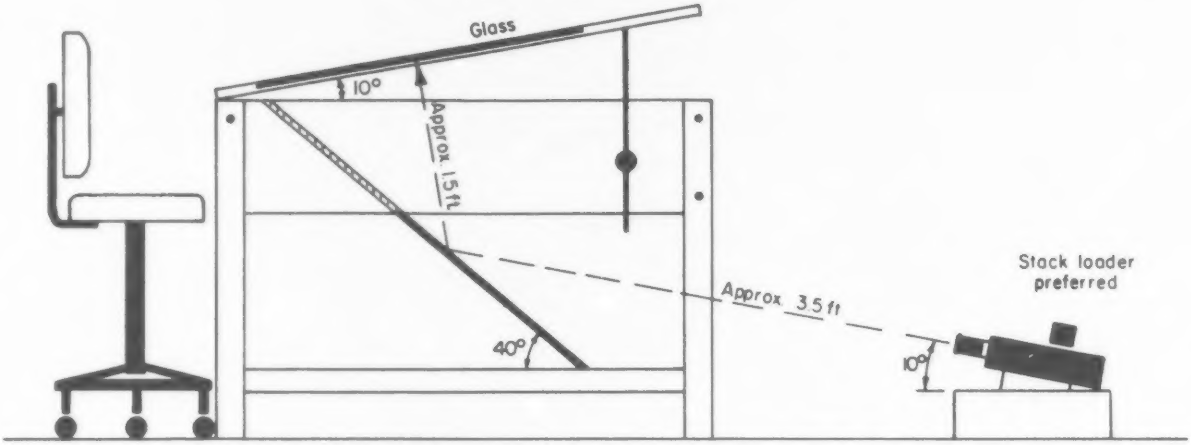

JOINT ROUGHNESS PROFILE
1
0-2
2
2-4
3
4-6
4
6-8
5
8-10
6
10-12
7
12-14
8
14-16
9
16-18
10
18-20

(after Barton 1977)

PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

# CORE SLIDE PROJECTION LAYOUT

Figure **4.6**


Glass
10°
Approx 1.5 ft
Stack loader preferred
Approx 3.5 ft
40°
10°

STURDY DRAFTING TABLE

Kodak Carousel 750H
w/120 mm lens

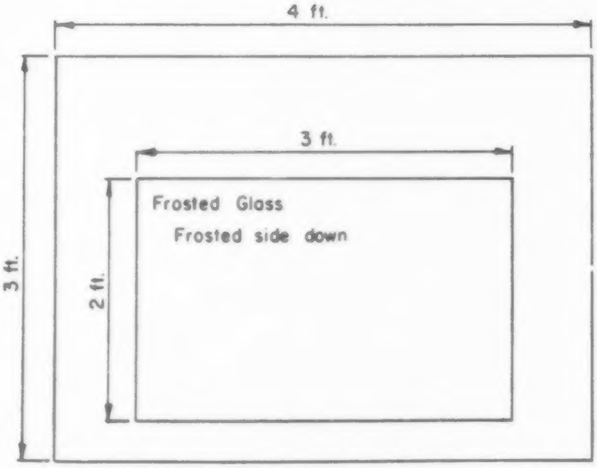
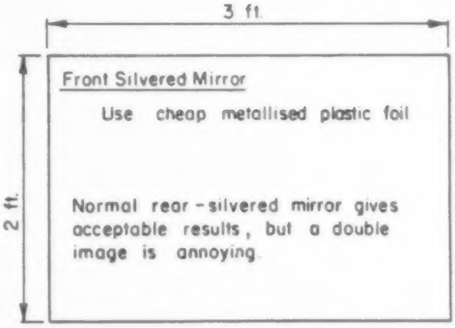

4 ft.
3 ft.
3 ft.
2 ft.
Frosted Glass
Frosted side down

TABLE TOP



Front Silvered Mirror

Use cheap metallised plastic foil

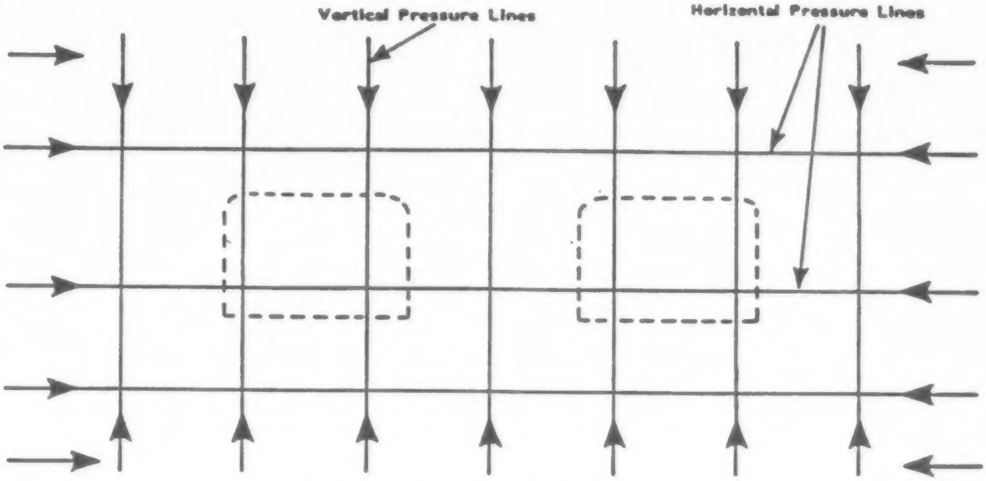
Normal rear-silvered mirror gives acceptable results, but a double image is annoying.

MIRROR

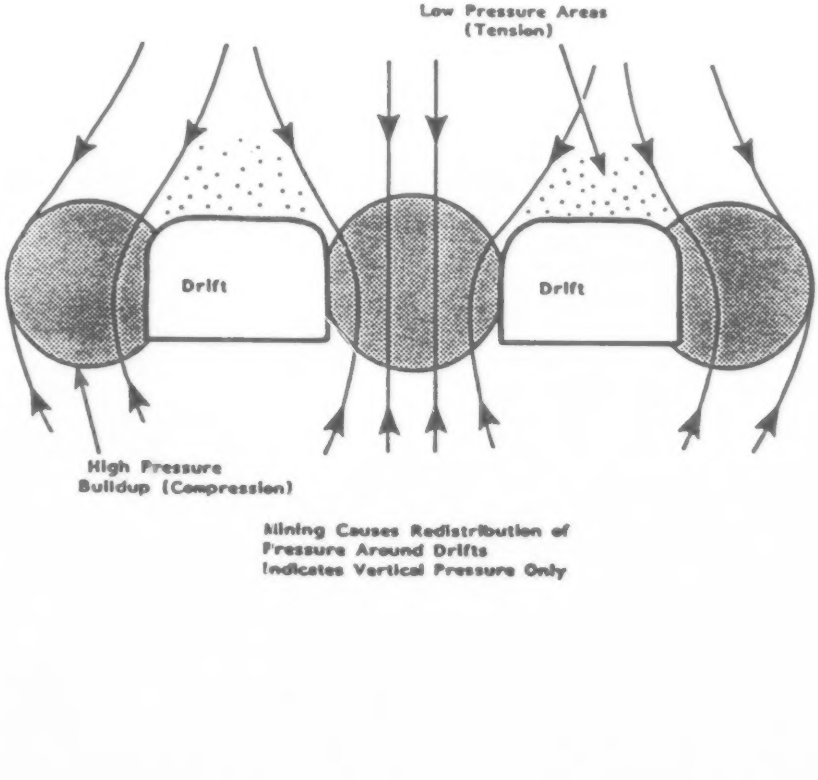
PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

# STRESS REDISTRIBUTION

Figure **5.1**


Vertical Pressure Lines
Horizontal Pressure Lines

No Mining – Even Pressure Distribution

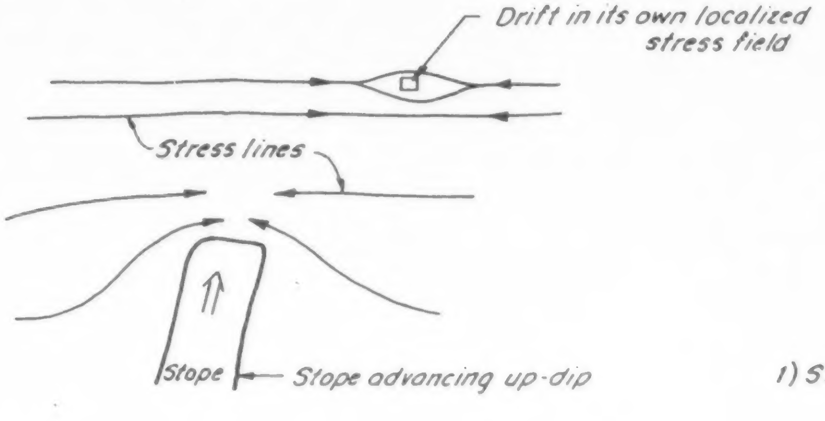

Low Pressure Areas
(Tension)
Drift
Drift
High Pressure
Buildup (Compression)

Mining Causes Redistribution of
Pressure Around Drifts
Indicates Vertical Pressure Only

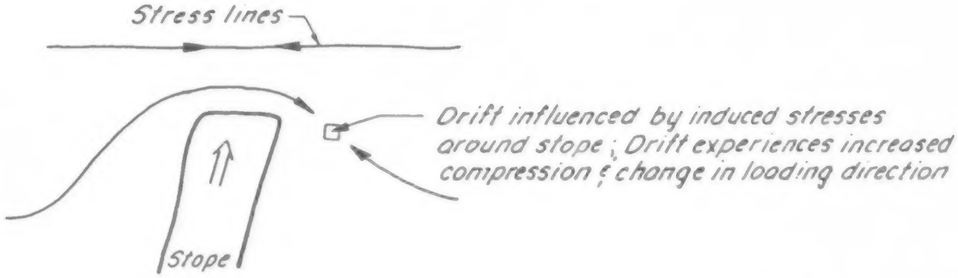
PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

## STESS CHANGES DUE TO MINING

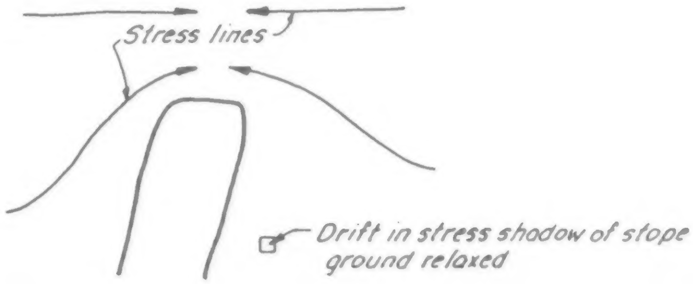
Figure **5.2**


Drift in its own localized stress field
Stress lines
Stope
Stope advancing up-dip

1) STOPE REMOTE FROM DRIFT


Stress lines
Drift influenced by induced stresses around stope ; Drift experiences increased compression & change in loading direction
Stope

2) BACK OF STOPE SAME ELEVATION AS DRIFT


Stress lines
Drift in stress shadow of stope ground relaxed

3) STOPE PASSED DRIFT

PROJECT NO. .......... DRAWN .......... REVIEWED .......... DATE ..........

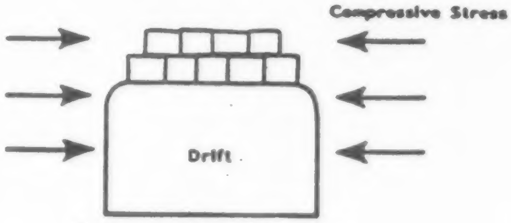
# FAILURE DUE TO GROUND RELAXATION

Figure **5.3**

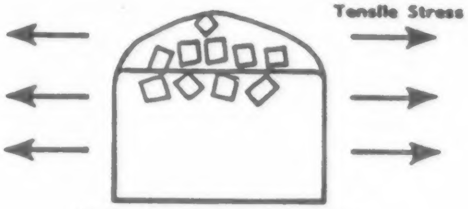
Changes in pressure can lead to ground problems.

Rock generally has low strength in tension because of the presence of weakness planes and therefore this is one of the most frequent causes of failures underground.

Figure A shows the positive contribution to stability of horizontal pressures and illustrates tensile failure.

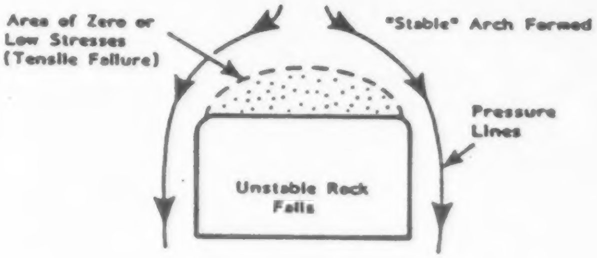

Compressive Stress
Drift

Series of blocks held together by horizontal forces


Tensile Stress

Removal of compressive stresses or presence of tensile stresses causes blocks to crumble and collapse

Illustration of Tensile Failure


Area of Zero or Low Stresses (Tensile Failure)
"Stable" Arch Formed
Pressure Lines
Unstable Rock Falls

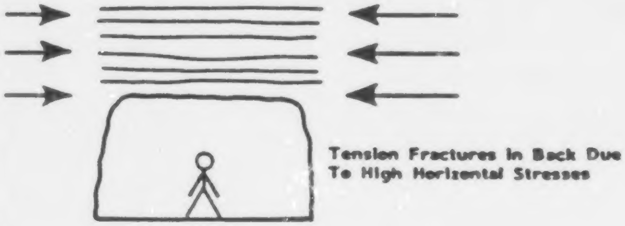
Example of Tensile Failure

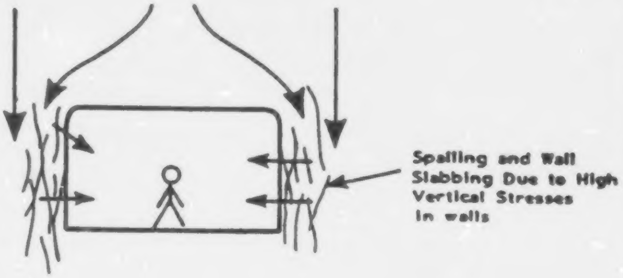
PROJECT NO. ............ DRAWN ......... REVIEWED ......... DATE ...........

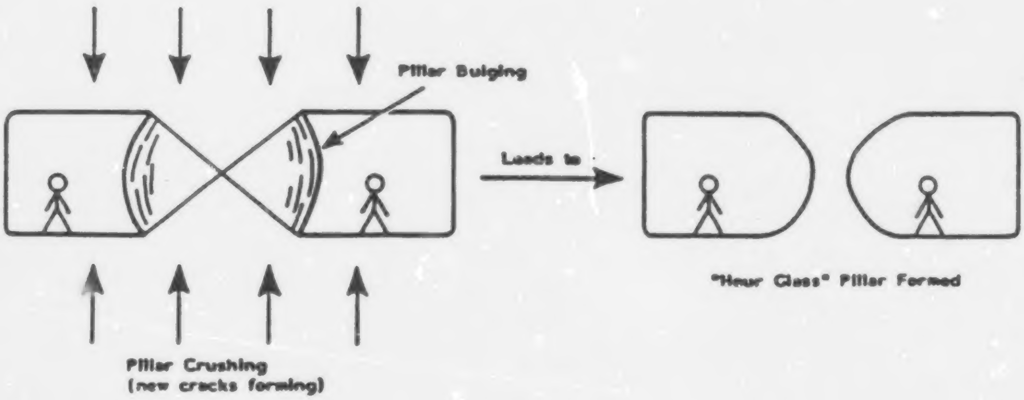
# FAILURE DUE TO HIGH COMPRESSIVE LOADS

Figure **5.4**

Rock failure due to very high pressures are generally characterized by thin slabbing off pillar walls, horizontal fracturing in stope backs or crushed, "hour-glassing" of pillars.


Tension Fractures In Back Due
To High Horizontal Stresses

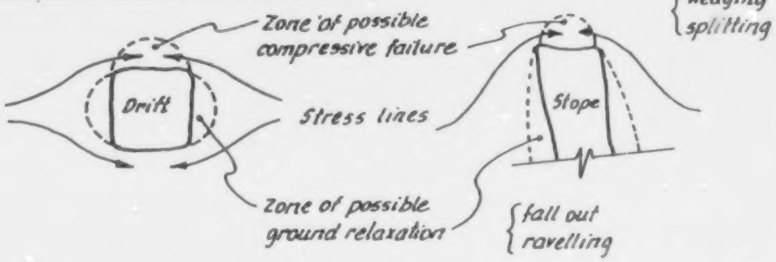

Spalling and Wall
Slabbing Due to High
Vertical Stresses
in walls


Pillar Bulging
Leads to
Pillar Crushing
(new cracks forming)
"Hour Glass" Pillar Formed

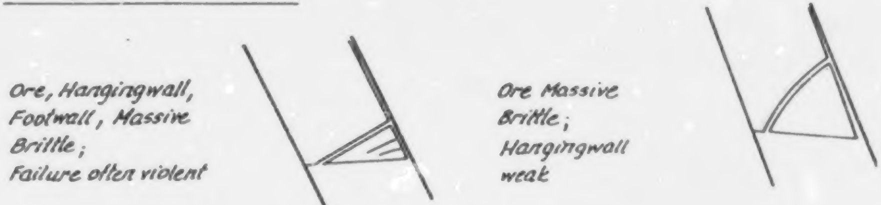
(From OML, 1986)

PROJECT NO. .............. DRAWN .............. REVIEWED .............. DATE ..............

## 1. APPROXIMATE STRESS DISTRIBUTION

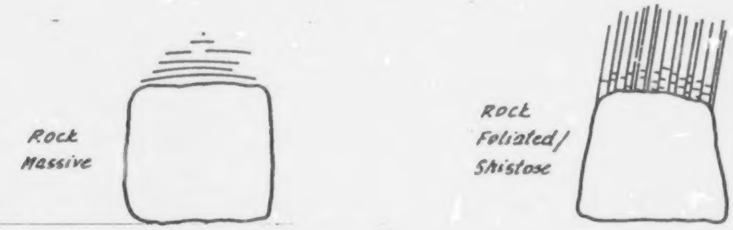

Zone of possible compressive failure
Drift
Stope
Stress lines
Zone of possible ground relaxation
wedging
splitting
fall out
ravelling

## 2. ROOF/BACK FAILURES


Ore, Hangingwall, Footwall, Massive Brittle; Failure often violent

Ore Massive Brittle; Hangingwall weak

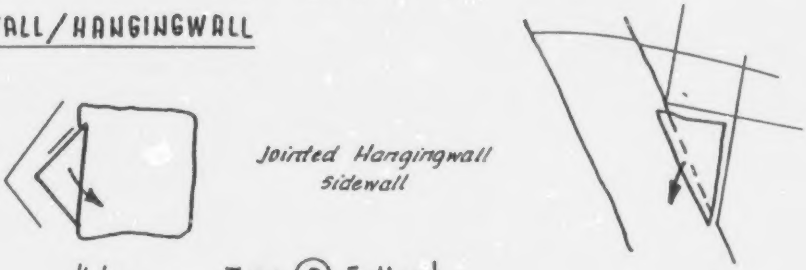
Type Ⓐ Wedging


Rock Massive

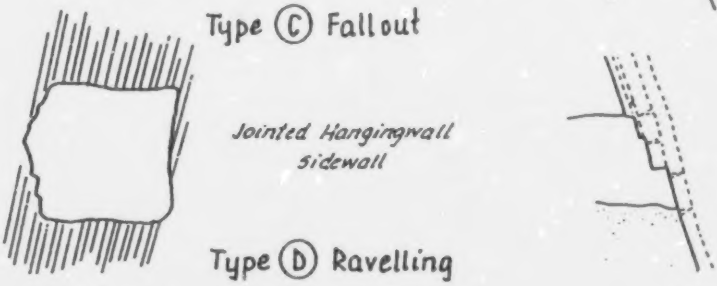
Rock Foliated/ Shistose

Type Ⓑ Splitting

## 3. SIDEWALL/HANGINGWALL


Jointed Hangingwall Sidewall

Type Ⓒ Fallout


Jointed Hangingwall Sidewall

Type Ⓓ Ravelling

PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

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## 1. APPROXIMATE STRESS DISTR

Drift
Stress lines
Failure

## 2. ROOF/BACK FAILURES

Type

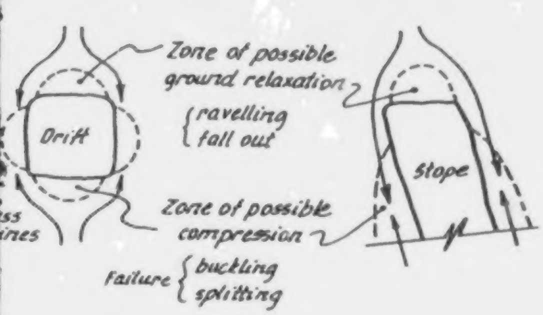
Foliated Rock

Type

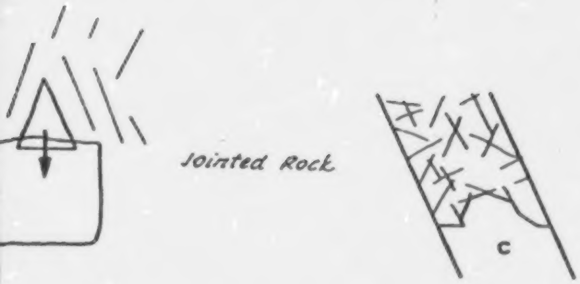
# INDICATORS OF HIGH COMPRESSIVE LOADS

**Figure 5.5**

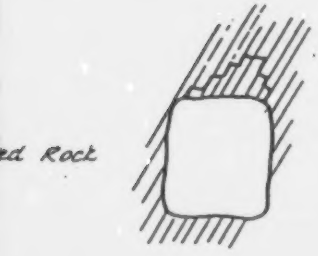
STRESS DISTRIBUTION


Zone of possible ground relaxation
ravelling
fall out
Drift
Stope
ss
ines
Zone of possible compression
Failure
buckling
splitting

ILURES

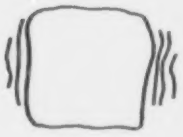

Jointed Rock
c

Type ① Fallout

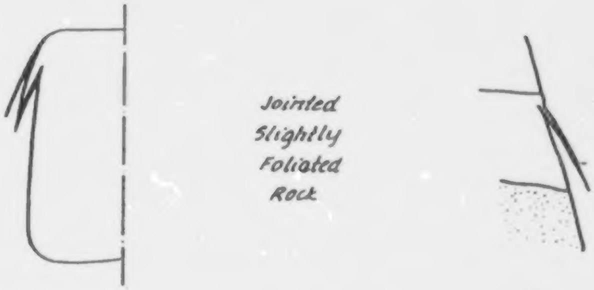


Type ② Ravelling

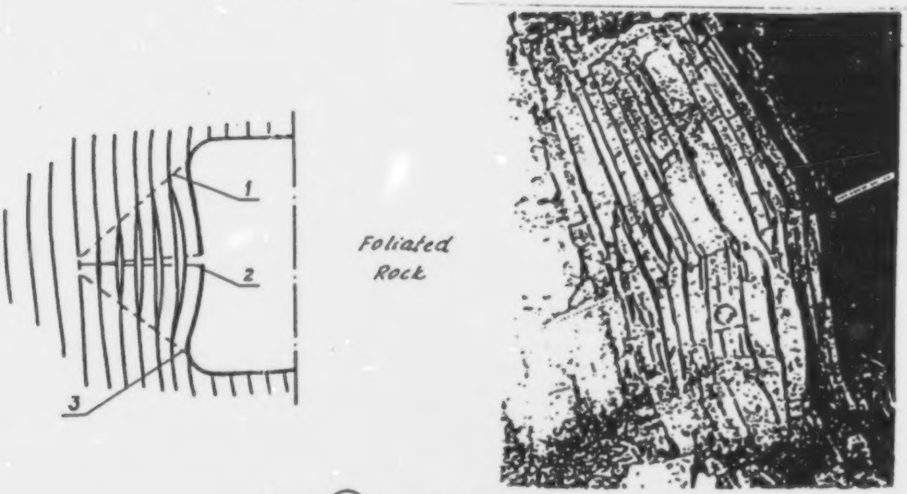
3. SIDEWALL / HANGINGWALL


Massive Rock

Type ③ Splitting


Jointed
Slightly
Foliated
Rock

Type ④ Shear


1
2
3
Foliated
Rock

Type ⑤ Buckling

# OBSERVATION SHEET - OPENINGS

Figure **5.6**

LOCATION: ____________________ SHEET ____ OF ____

DATE: __________ LEVEL: __________ STOPE: __________

N^o ON PLAN: ________ DISTANCE FW DRIFT TO STOPE: __________

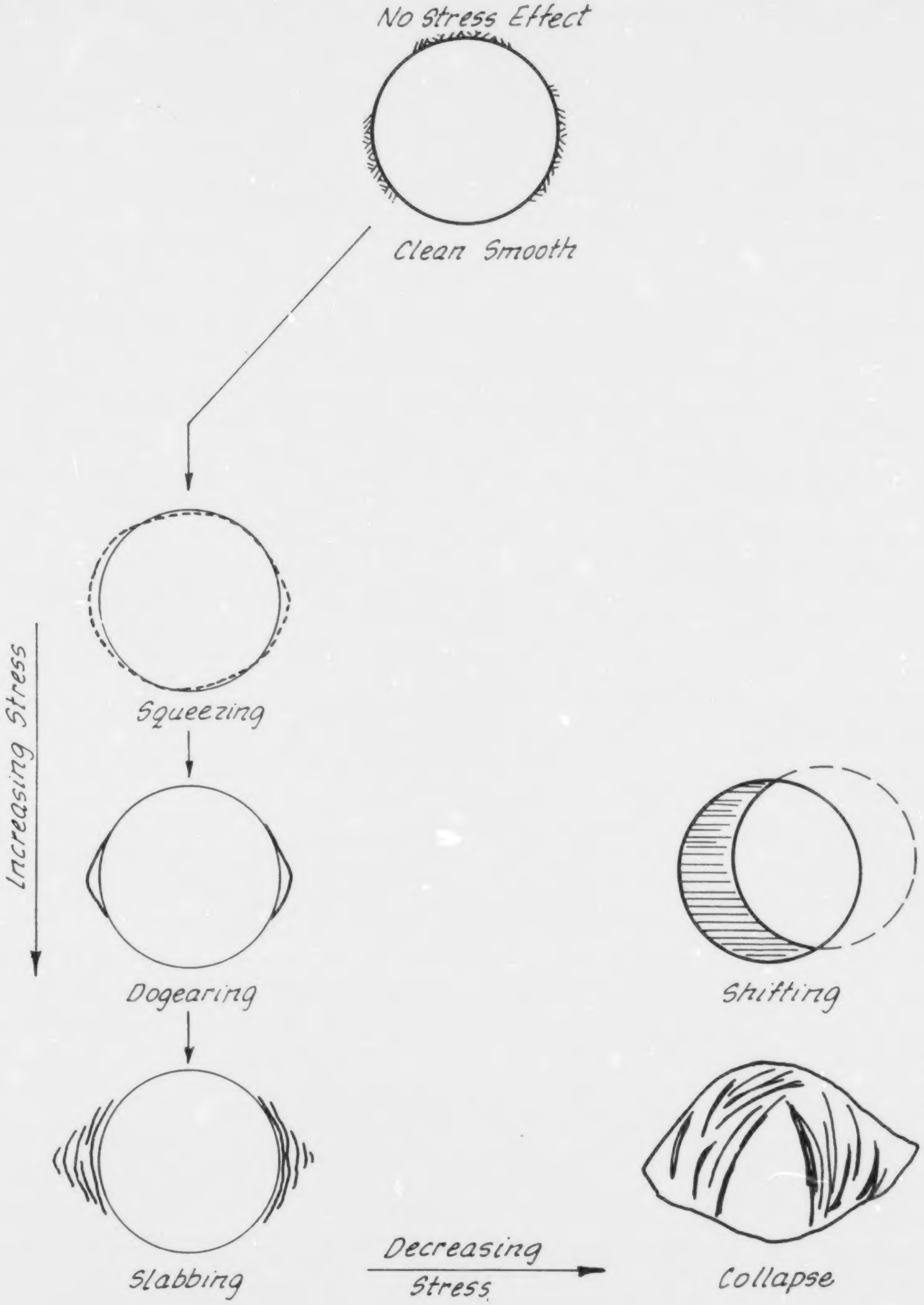
EXCAVATION CONDITION:

| | | STRESS STATE | | | | | SUPPORT | | COMMENTS |
|---|---|---|---|---|---|---|---|---|---|
| EXCAVATION | SURFACE | LOOSE | NO EVIDENCE | SLIGHT | MODERATE | SEVERE | TYPE | LOAD | |
| FOOTWALL HAULAGE | HW | | | | | | | | |
| | FW | | | | | | | | |
| | BACK | | | | | | | | |
| STOPE | HW | | | | | | | | |
| | FW | | | | | | | | |
| | BACK | | | | | | | | |
| | ENDWALL | | | | | | | | |
| DRILL DRIFT | HW | | | | | | | | |
| | FW | | | | | | | | |
| | BACK | | | | | | | | |
| EXTRACTION DRIFT | HW | | | | | | | | |
| | FW | | | | | | | | |
| | BACK | | | | | | | | |

PROJECT NO ........ DRAWN ........ REVIEWED ........ DATE ........

**BLASTHOLE CONDITION**

Figure **5.7**


No Stress Effect
Clean Smooth
Squeezing
Increasing Stress
Dogearing
Slabbing
Shifting
Decreasing Stress
Collapse

PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

## I.T.H. OBSERVATION

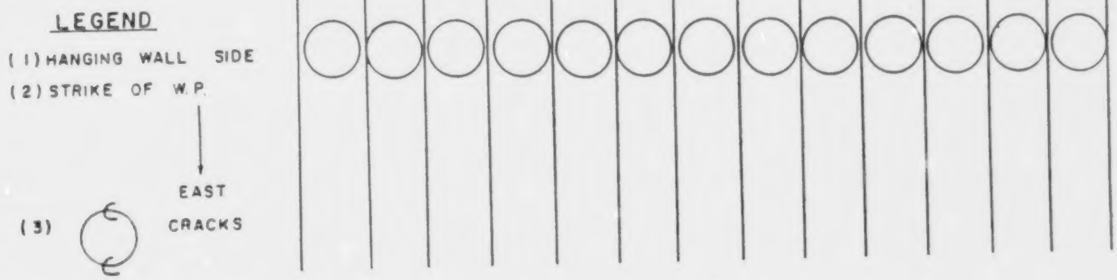
Figure **5.8**

I.T.H. OBSERVATIC __

DATE: ________ ______

LEVEL ______ W. P. ______ BLOCK** ____ ROW** _____ HOLE DIA. _____ in.

| | HOLE NUMBER / FT. FROM COLLAR | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| BLAST DATE ________<br>BLAST TYPE ________<br>BLAST NUMBER ________ | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 |
| HOLE POSITION TO FACE ___ FT. | | | | | | | | | | | | | |
| VERTICAL | | | | | | | | | | | | | |
| HORIZONTAL | | | | | | | | | | | | | |
| HOLE CONDITION: CLEAN/SMOOTH | | | | | | | | | | | | | |
| OVAL | | | | | | | | | | | | | |
| STRESS | | | | | | | | | | | | | |
| SHEAR | | | | | | | | | | | | | |
| COLLAPSE | | | | | | | | | | | | | |
| TOP SILL: GOOD | | | | | | | | | | | | | |
| LIFTED | | | | | | | | | | | | | |
| CLOSED HOLES: POUND OPEN | | | | | | | | | | | | | |
| REDRILL | | | | | | | | | | | | | |


LEGEND
(1) HANGING WALL SIDE
(2) STRIKE OF W.P.
EAST
(3)
CRACKS

PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

PROJECT NO .................. DRAWN .......... REVIEWED .......... DATE ......................

# BOREHOLE RECORD

Figure 6.1

```
AW600B                      BOREHOLE RECORD          DATE PROCESSED 06/13/[illegible]               ASSAYS CHK'D.......

                                                                                                    DATE.................
BOREHOLE     MINE   NUMBER LEVEL  DEPTH AZIMUTH  DIP  CO ORD  LATITUDE   DEPARTURE   ELEVATION   STARTED COMPLETED  CMPLT MRGD
 74502 0 THOMPSON     63 2800.     90.   90 0   62 0    1    N [illegible]  E 1824    3101.      3  6 87   3 11 87     1     1

                                       T R O P A R I    T E S T S

    DEPTH     AZIMUTH   DIP       DEPTH     AZIMUTH   DIP      DEPTH     AZIMUTH   DIP       DEPTH     AZIMUTH   DIP
     10       90  0   +62  0       85       90  0   +63 30       0        0  0    0 0  0       0        0  0    0 0  0


LOGGED BY K SMITH,EXT INCO,30 ST,CHECKED BY N I EKESTRAND                                                    PROGRESSIVE TOTALS
                                 MINERALIZATION
 DEPTH LENGTH  SAMPLE    EST.G  CU     NI    CO   CU+NI    S     SG    TEM   ORE ROCK       DESCRIPTION           CU      NI    CU+NI
   0.0   0.0                                                                            COLLAR
  29.5  29.5                                                                       II    DK GY GRN TO BLK,QTZ,AMPB,GAR,
                                                                                       FG,OCC PK
                                                                                       CG PEG BNDS,BNDD,45-55
                                                                                       R4,JN2,JR2,JA0,RQD 055
  49.0  19.5                                                                      SCH   BIOT,SLCS,OCC MG WHT PEG INCS,
                                                                                       BLKY,BNDD
                                                                                       55-50
                                                                                       R3,JN2,JR2,JA0,RQD 025
  54.5   5.5   600038    3.50  0.21  2.21 0.04  2.42  0.0   3.41        SUMX          50%,PO PN PY,2XB2,FG,SCH INCS,   1.2    12.2    13.3
                                                                                       BLKY
                                                                                       R3,JN2,JR5,JA0,RQD 010
  62.4   7.9   600039 * 0.40  0.16  1.04 0.02  1.20  0.0   3.18        MSCH          5%,SULP STRS,FG,BLKY,BNDD,45-3   2.4    20.4    22.8
                                                                                       5
                                                                                       R3,JN3,JR2,JA0,RQD 015
  74.0  11.6   600040    0.20  0.06  0.33 0.01  0.39  0.0   3.04              SCH    BIOT,GY,FG,BLKY,BNDD,35-45-40    3.1    24.2    27.3
                                                                                       R3,JN2,JR2,JA0,RQD 015
  79.0   5.0   600041    2.80  0.08  2.61 0.05  2.69  0.0   3.49        SUMX          40%,PO PN PY,2XA2,FG,NMRS SCH    3.5    37.2    40.8
                                                                                       INCS
                                                                                       R3,JN2,JR5,JA0,RQD 050
  90.0  11.0                                                                      SCH   BIOT,GY,FG,BLKY,BNDD,20-25
                                                                                       R3,JN4,JR2,JA2,RQD 015
                                                                                       FOOT OF HOLE
```

PROJECT NO .................. DRAWN ........ REVIEWED ........ DATE ....................

## ROCKMASS QUALITY

Figure **6.2**

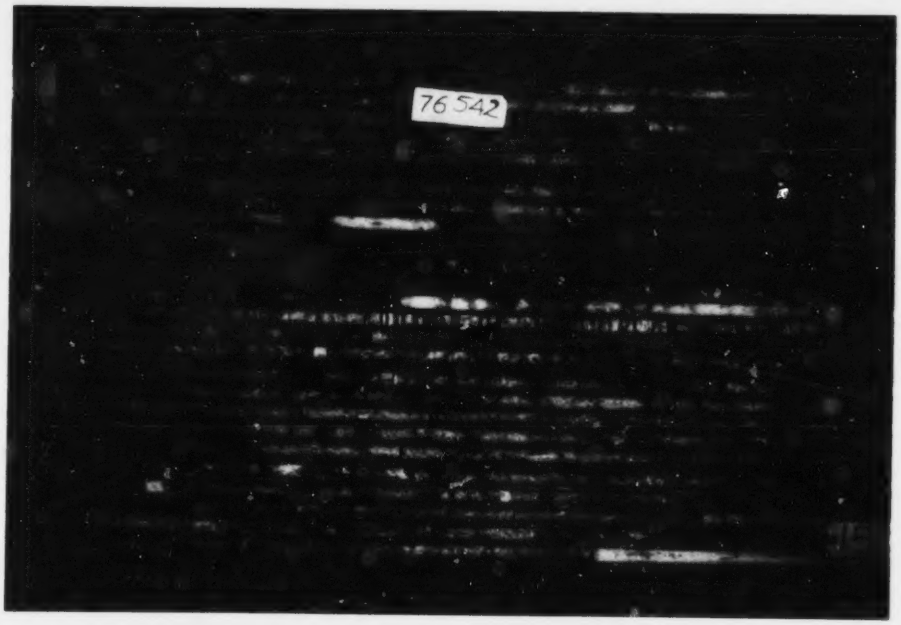
```
BHRTRP40          ROCK MASS QUALITY                         88-06-13

BOREHOLE   745020                 INTERVAL FROM:          TO 99999.9
 DEPTH  LENGTH  ROCK #RQD #JN #JR    #JA    #JW   #SRF  #Q
 -----  ------  ----  ----  ---  ---   ----   ----   ----  ----

                         *** INTERVAL START ***
   29.5    29.5 IF     55   9  1.5  0.75  1.00  1.0   12
   49.0    19.5 SCH    25   9  1.5  0.75  1.00  1.0    5
   54.5     5.5        10   9  3.0  0.75  1.00  1.0    4
   62.4     7.9        15   9  1.5  0.75  1.00  1.0    3
   74.0    11.6 SCH    15   9  1.5  0.75  1.00  1.0    3
   79.0     5.0        50   9  3.0  0.75  1.00  1.0   22
   90.0    11.0 SCH    15   9  1.5  2.00  1.00  1.0    1
          ------     ----  *** INTERVAL END   ***    ----
AVERAGE    90.0      31.9                            7.3
   90.0 *****END OF BOREHOLE
```

# P.D. DRILL CORE PHOTOGRAPHY

FIGURE 6.3



# ROCK MASS CLASSIFICATION DATA SHEET

Figure 6.4

LOCATION: Level 1600 Face ______ Distance ______

DATE 20 Jan 89

| | | |
|---|---|---|
| Rock Type | Strength | Sketch |
| Fw Schist | R3 | |

| Joint Data | Set 1 | Set 2 | Set 3 | Set 4 |
|---|---|---|---|---|
| Dip Direct/Dip Angle | 135/65 | 190/85 | NOT OBSERVED | — |
| Average Spacing (ft) | 0.1 | 5 | - | — |
| Estimated No. of Joints/$m^3$ | | | | |

Q' = 1.1

MRMR =

Avg. Vol. =

Shape

RQD ~ 30

## JOINT ALTERNATION NUMBER (Ja)

| Description of gouge | Joint alteration number (Ja) for joint separation (mm) | | |
|---|---|---|---|
| | <1.01[1] | 1.0-5.0[2] | >5.0[3] |
| Tightly healed, hard, nonsoftening impermeable filling | 0.75 | - | - |
| Unaltered joint walls, surface staining only | 1.0 | - | - |
| Slightly altered, non-softening, non-cohesive rock mineral or crushed rock filling | 1. [✓ 2.0] | 4.0 | 6.0 |
| Non-softening, slightly clayey non-cohesive filling | 3.0 | 2. [✓ 6.0] | 10.0 |
| Softening or low friction clay mineral coatings and small quantities of swelling clays | 4.0 | 8.0[5] | 13.0 |
| Shattered or micro-shattered (swelling) clay gouge, with or without crushed rock | 5.0 | 10.0[4] | 18.0 |

## JOINT ROUGHNESS NUMBER (Jr)

| Joint Separation | Condition of Joint | Joint Roughness Number (Jr) |
|---|---|---|
| Joints/fissures tight or losing during excavation | Discontinuous joint/fissures | [✓ 4.0] 1. |
| | Rough or irregular, indulating | 3.0 |
| | Smooth indulating | 2.0 |
| | Rough or irregular, planar | 1.5 |
| | Smooth planar | [✓ 1.0] 2. |
| | Slickensided planar | 0.5 |
| Joints/fissures open and remain open during excavation | Joints/fissures either open or containing relatively soft gouge or sufficient thickness to prevent joint/fissure wall contact upon excavation. | 1.0 |
| | Shattered or micro-shattered clays | 1.0 |

## STRESS REDUCTION FACTOR (SRF)

| | |
|---|---|
| Weakness zones intersecting pillar. | 10.0 |
| Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock (any depth). | 2.5 |
| Single weakness zones containing clay, or chemically disintegrated rock. | 2.5 |
| Multiple shear zones in competent rock (clay free), loose surrounding rock. | 7.5 |
| Single shear zones in competent rock (clay free). | 2.5 |
| Loose open joints, heavily jointed or 'sugar cube'. | 5.0 |

## JOINT SET NUMBER (Jn)

| Number of joint sets | Joint set number (Jn) |
|---|---|
| Intact, no or few joint/fissures | 0.5 - 1.0 |
| One joint/fissure set | 2 |
| One joint/fissure set plus random | 3 |
| Two joint/fissure sets | 4 |
| Two joint/fissure sets plus random | 6 |
| Three joint/fissure sets | [✓ 9] |
| Three joint/fissure sets plus random | 12 |
| Four or more joint/fissure sets, random, heavily jointed, sugar cube, etc. | 15 |
| Multiple joint/fissure sets (crushed rock, earth-like) | 20 |

PROJECT NO.

DRAWN

REVIEWED

DATE

**Figure 6.5**

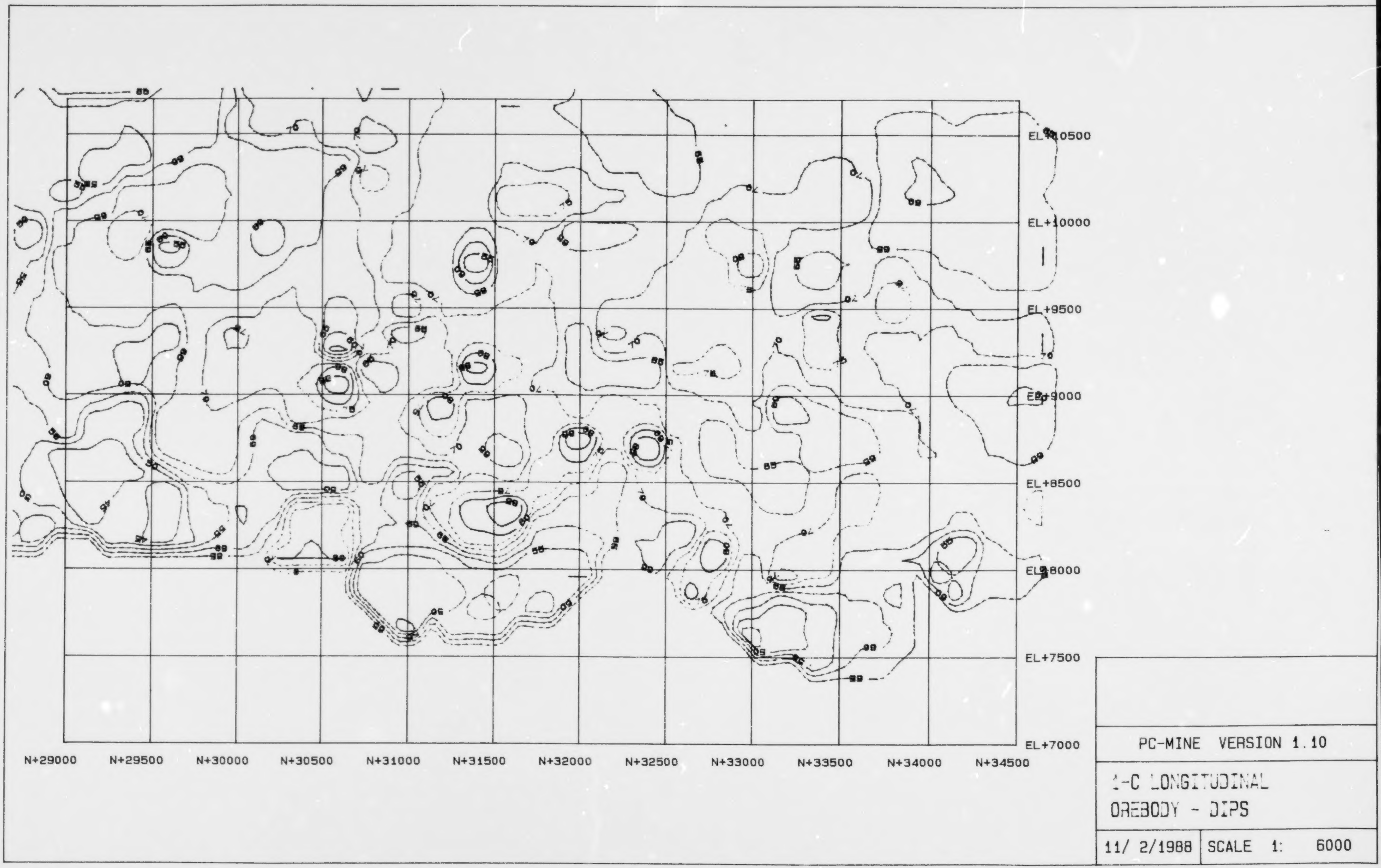
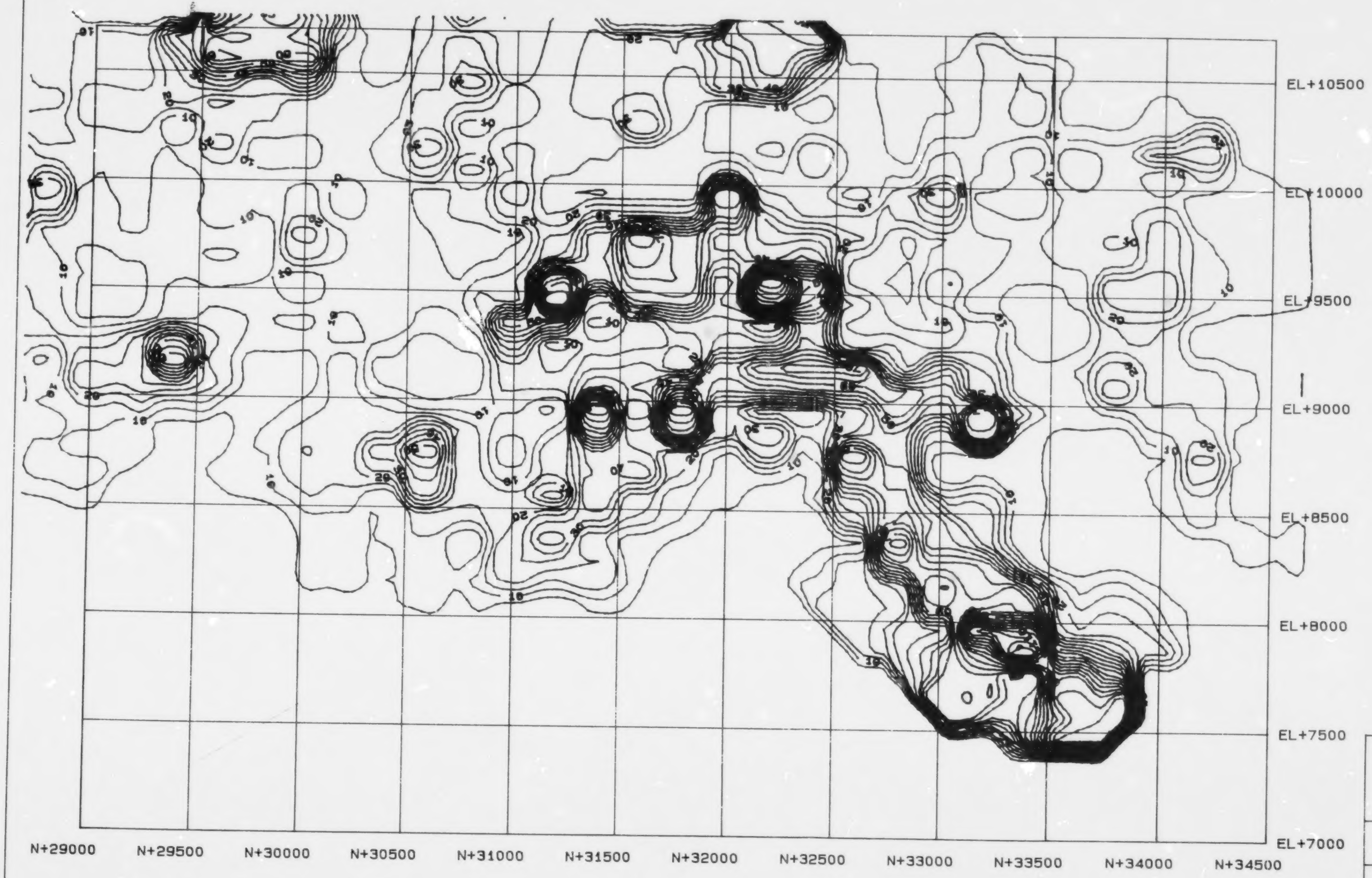
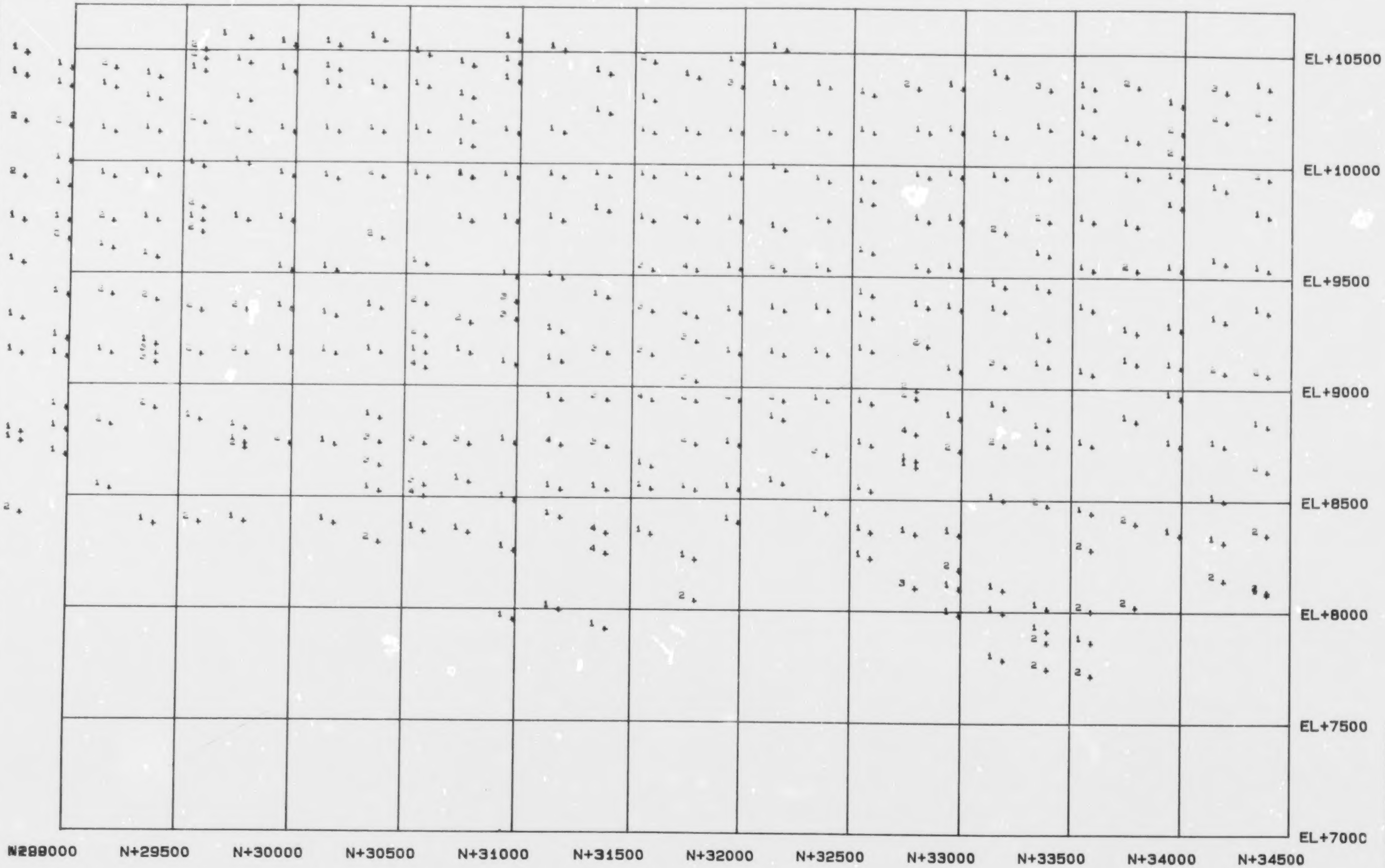

EL+10500
EL+10000
EL+9500
EL+9000
EL+8500
EL+8000
EL+7500
EL+7000
N+29000
N+29500
N+30000
N+30500
N+31000
N+31500
N+32000
N+32500
N+33000
N+33500
N+34000
N+34500
PC-MINE VERSION 1.10
1-C LONGITUDINAL
OREBODY - DIPS
11/ 2/1988
SCALE 1: 6000

**Figure 6.6**


EL+10500
EL+10000
EL+9500
EL+9000
EL+8500
EL+8000
EL+7500
EL+7000
N+29000
N+29500
N+30000
N+30500
N+31000
N+31500
N+32000
N+32500
N+33000
N+33500
N+34000
N+34500
10
20
30

| |
|---|
| PC-MINE VERSION 1.10 |
| 1-C LONGITUDINAL<br>OREBODY WIDTHS |
| 11/ 2/1988 SCALE 1: 6000 |

Figure 6.7



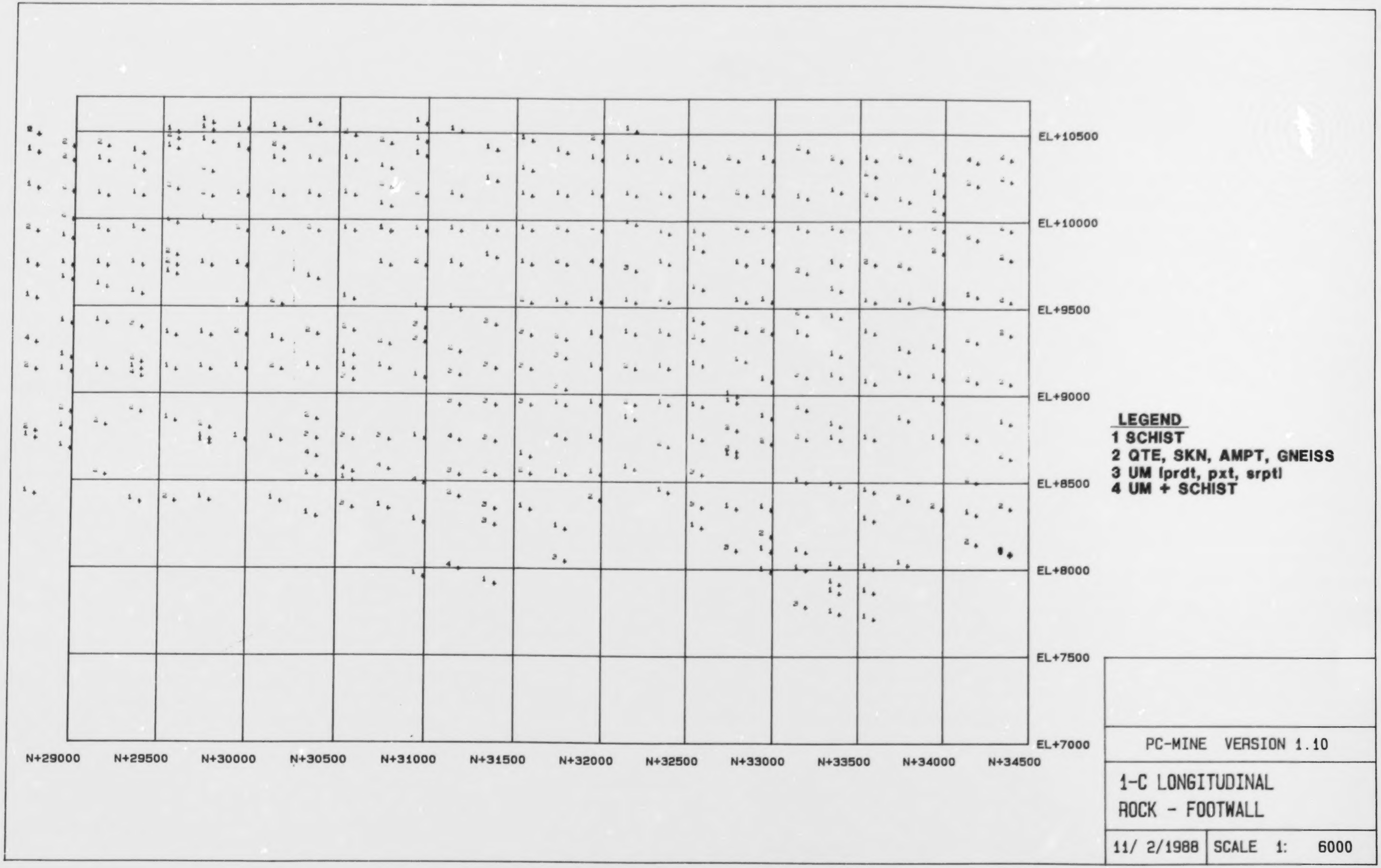
**LEGEND**
1 SCHIST
2 QTE, SKN, AMPT, GNEISS
3 UM (prdt, pxt, srpt)
4 UM + SCHIST

PC-MINE VERSION 1.10

1-C LONGITUDINAL
ROCK - HANGINGWALL

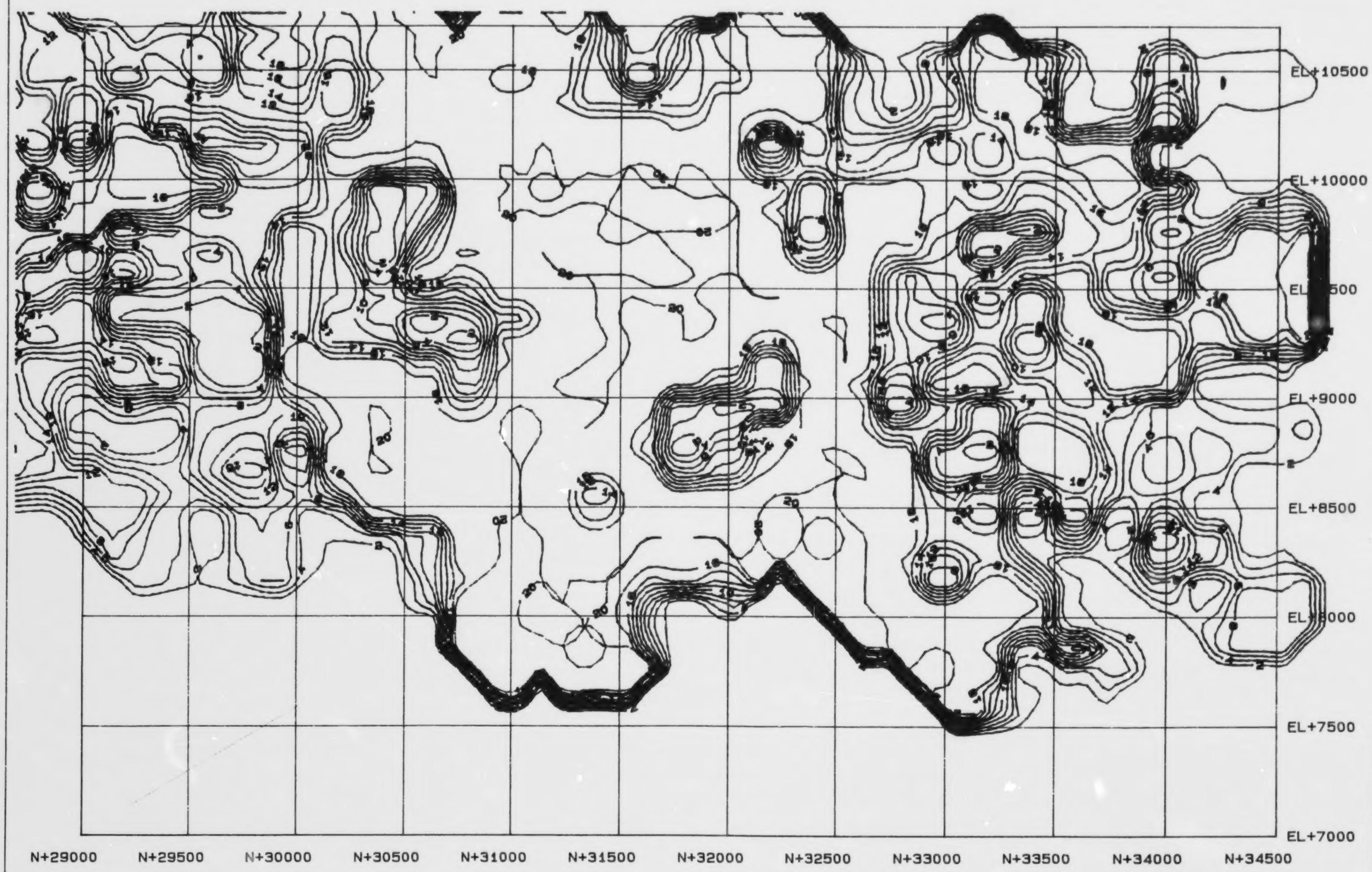
11/ 2/1988 SCALE 1: 6000

Figure 6.8



1-C LONGITUDINAL
ROCK - FOOTWALL

**Figure 6.9**

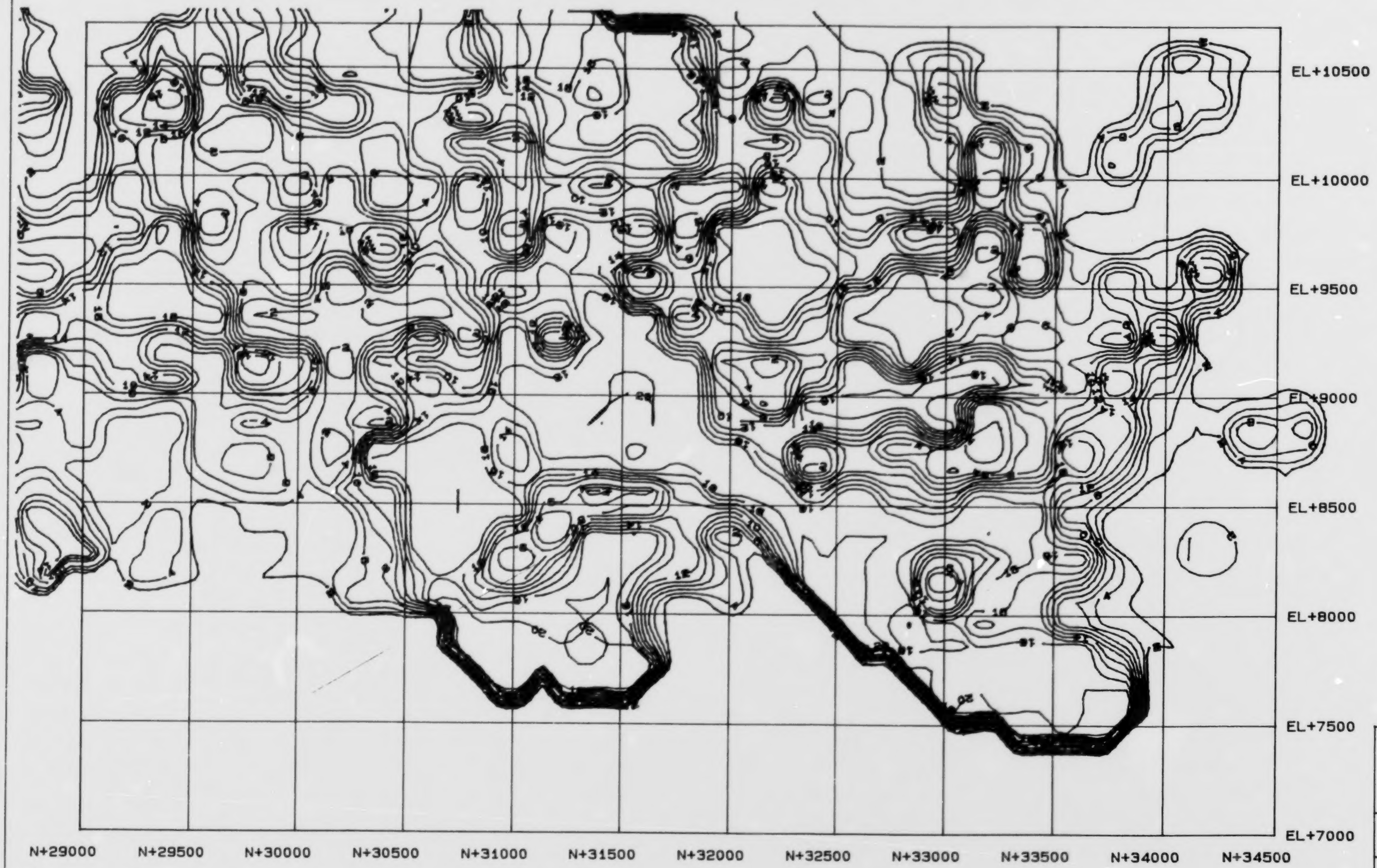

EL+10500
EL+10000
EL+9500
EL+9000
EL+8500
EL+8000
EL+7500
EL+7000
N+29000
N+29500
N+30000
N+30500
N+31000
N+31500
N+32000
N+32500
N+33000
N+33500
N+34000
N+34500

PC-MINE VERSION 1.10

1-C LONGITUDINAL
WIDTH - HANGINGWALL

| | |
|---|---|
| 11/ 2/1988 | SCALE 1: 6000 |

**Figure 6.10**

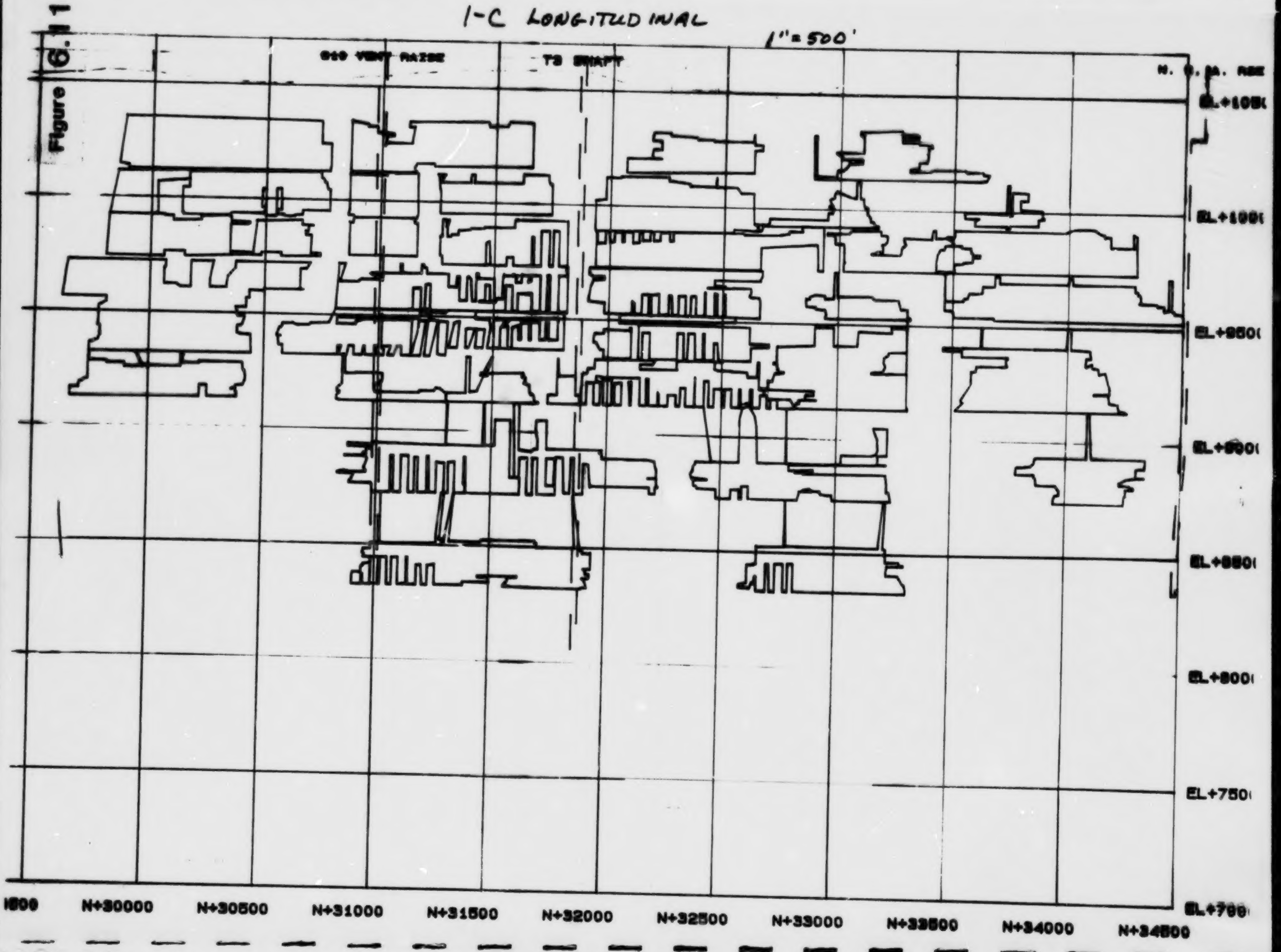

EL+10500
EL+10000
EL+9500
EL+9000
EL+8500
EL+8000
EL+7500
EL+7000
N+29000
N+29500
N+30000
N+30500
N+31000
N+31500
N+32000
N+32500
N+33000
N+33500
N+34000
N+34500

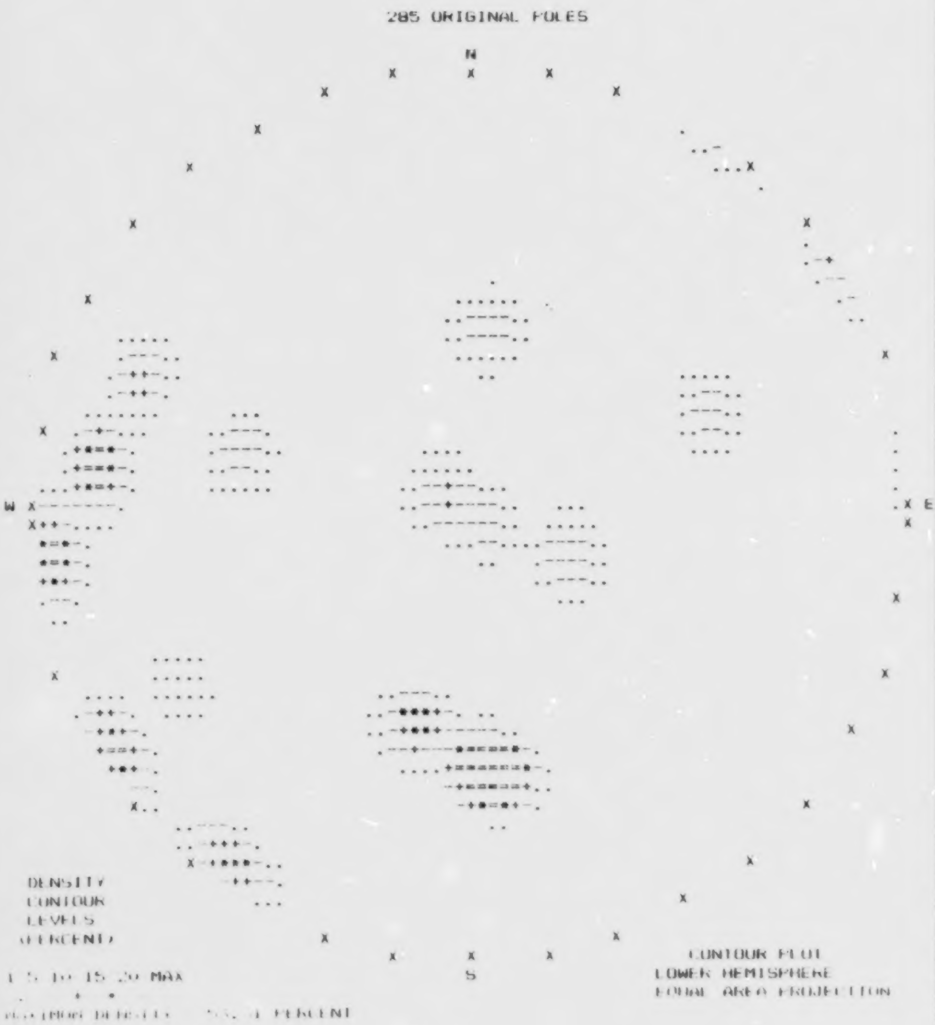
PC-MINE VERSION 1.10

1-C LONGITUDINAL
WIDTHS - FOOTWALL

11/ 2/1988 | SCALE 1: 6000

Figure 6.11


1-C LONGITUDINAL
1"=500'
810 VENT RAISE
T3 SHAFT
N. O. M. REE
EL+1050
EL+1000
EL+950
EL+900
EL+850
EL+800
EL+750
EL+700
500
N+30000
N+30500
N+31000
N+31500
N+32000
N+32500
N+33000
N+33500
N+34000
N+34500


285 ORIGINAL POLES
N
W
E
S
DENSITY
CONTOUR
LEVELS
(PERCENT)
1 5 10 15 20 MAX
CONTOUR PLOT
LOWER HEMISPHERE
EQUAL AREA PROJECTION
MAXIMUM DENSITY
PERCENT

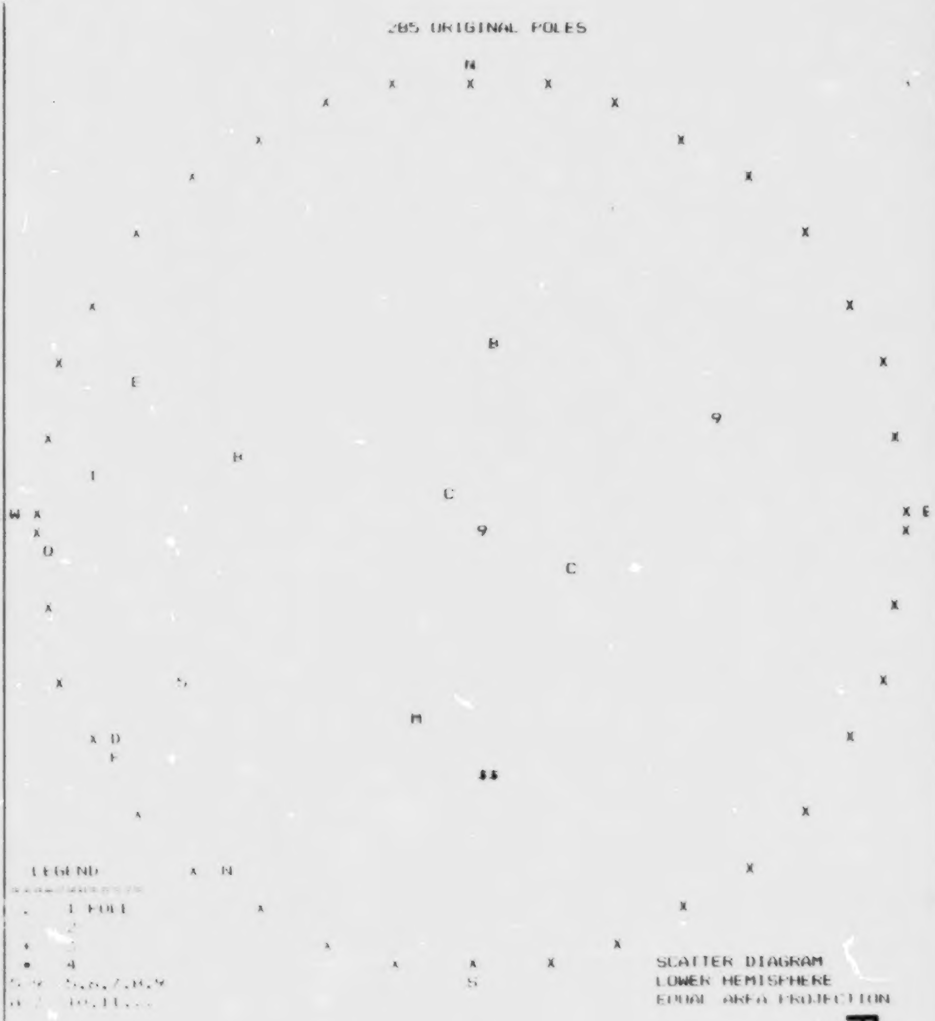
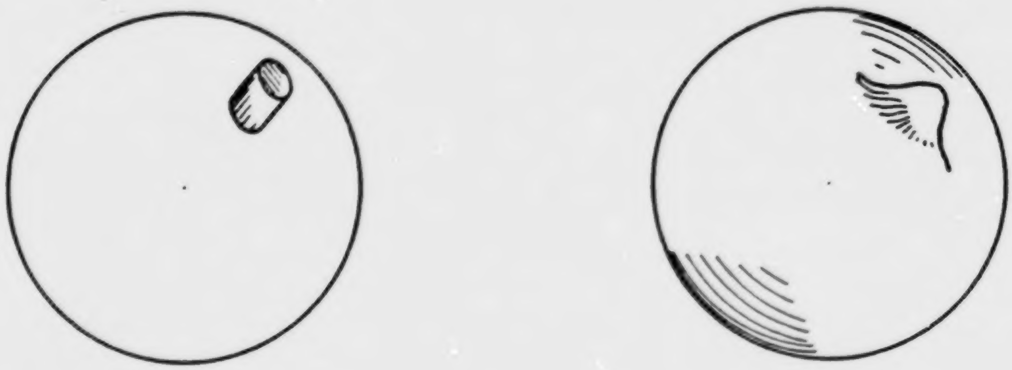

285 ORIGINAL POLES
N
W
E
S
LEGEND
1 POLE
2
3
4
5,6,7,8,9
10,11...
SCATTER DIAGRAM
LOWER HEMISPHERE
EQUAL AREA PROJECTION

Figure 6.12

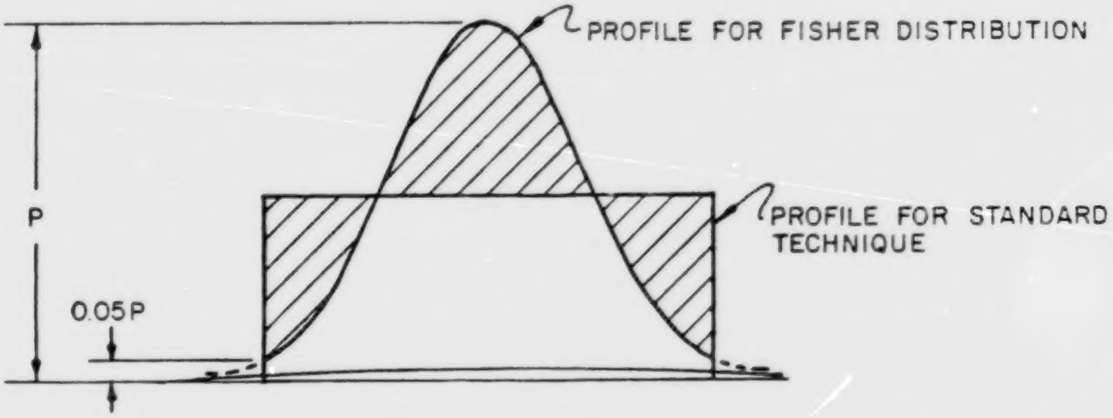
## COUNTING TECHNIQUES

Figure **6.13**



a) Density distribution of pole with standard counting technique

b) Fisher distribution of pole used in Golder technique

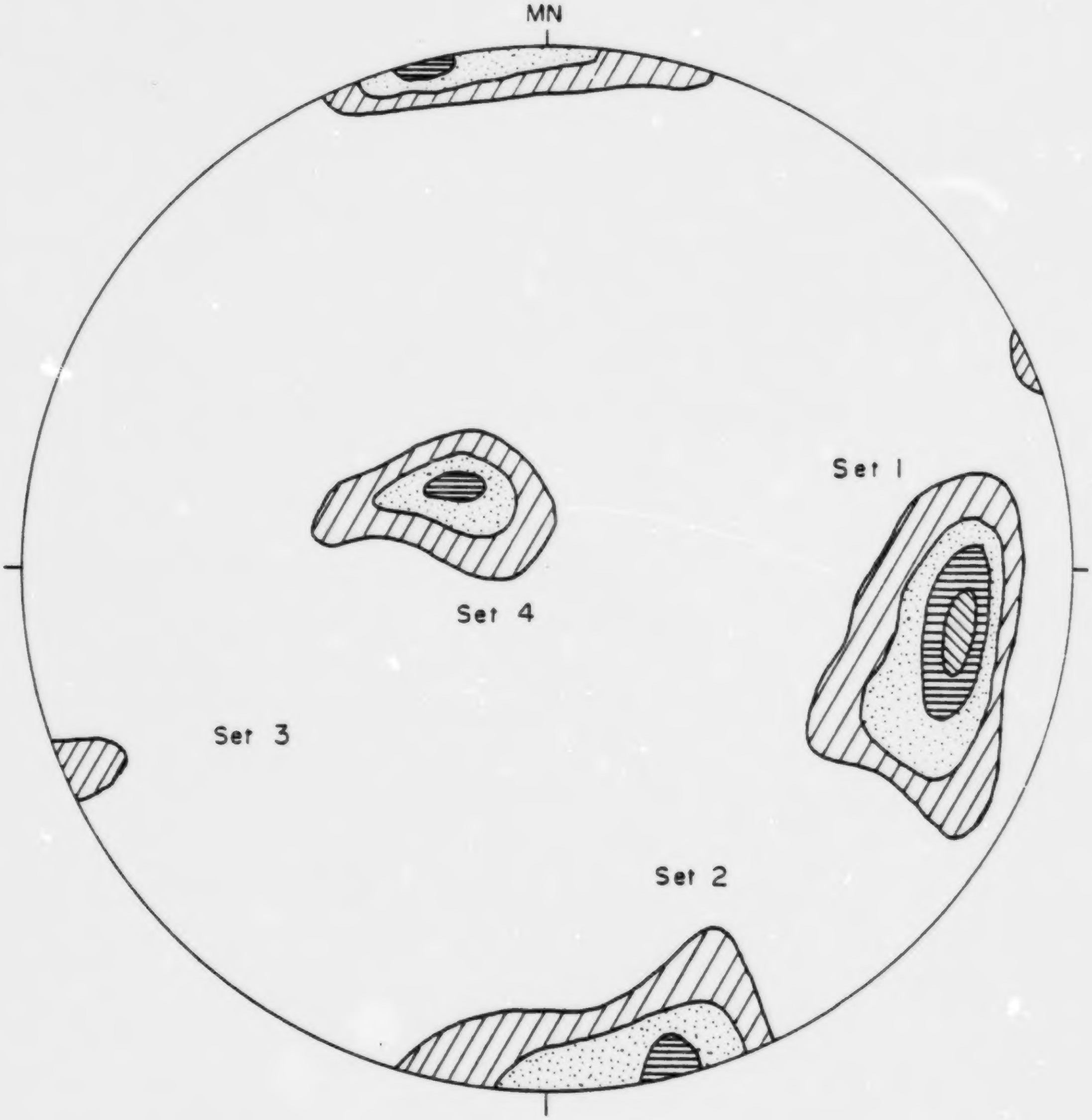

PROFILE FOR FISHER DISTRIBUTION
P
PROFILE FOR STANDARD TECHNIQUE
0.05P

c) Comparison of distribution profiles

PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

# STEREOPLOT OF FABRIC MAPPED ON 400 AND 1000 LEVELS

Figure **6.14**



UPPER HEMISPHERE
POLAR EQUAL AREA
STEREO-NET

MN ~ MINE NORTH

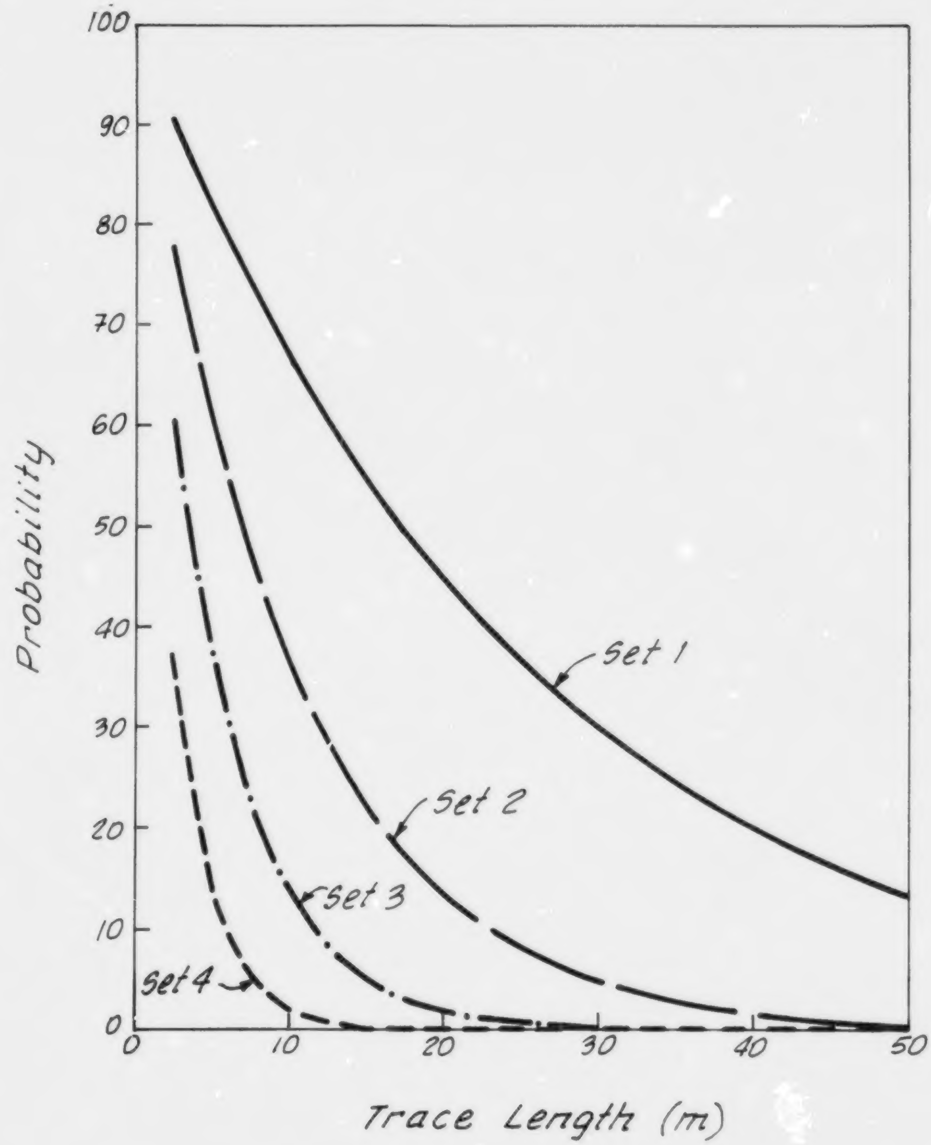
LEGEND

▧ 5%
▤ 4%
▒ 3%
▨ 2%

PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

## DISTRIBUTION OF JOINT TRACE LENGTH

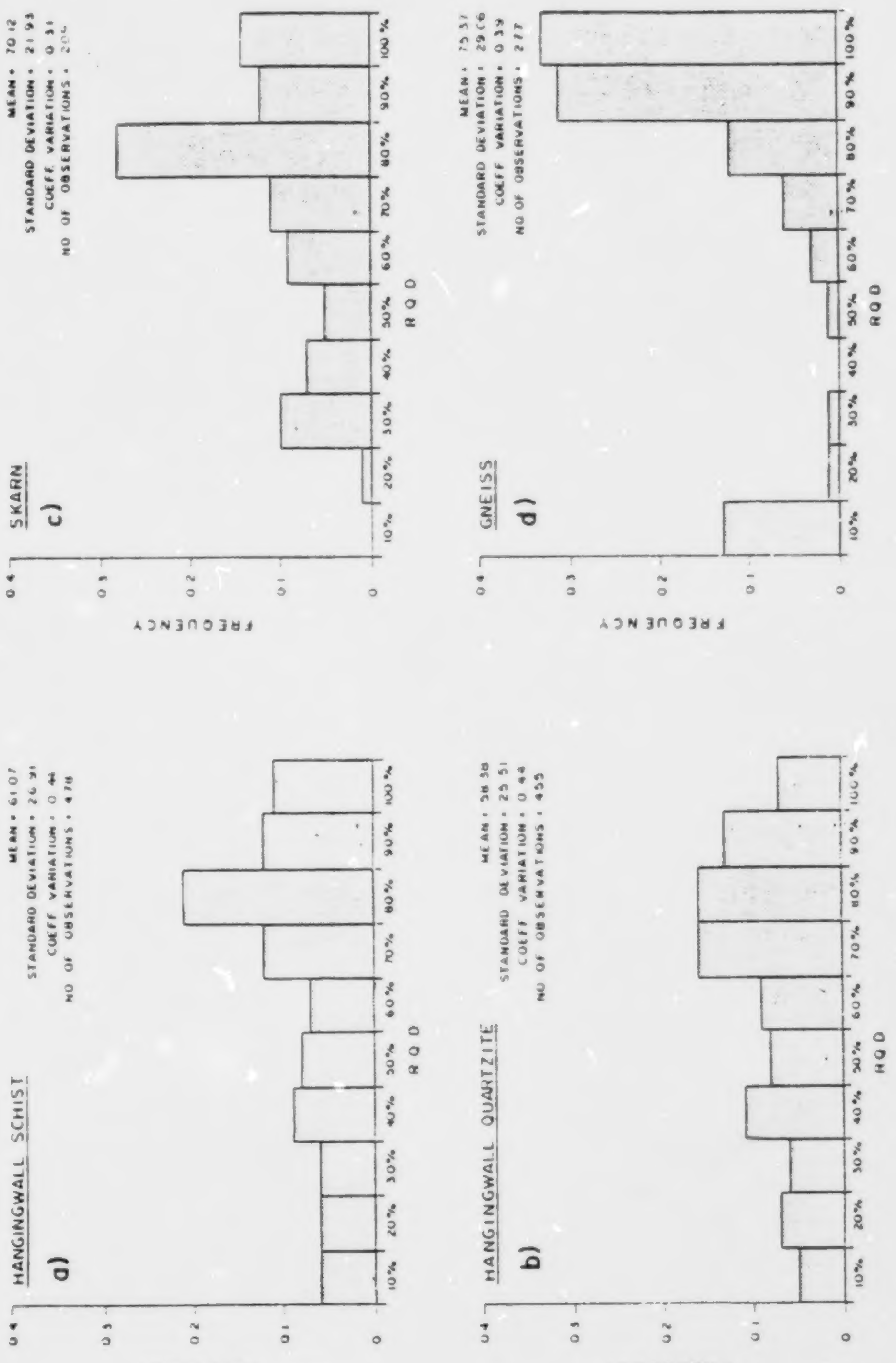
Figure **6.15**



PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

# HISTOGRAMS OF ROCK QUALITY DESIGNATION HANGINGWALL ROCKS

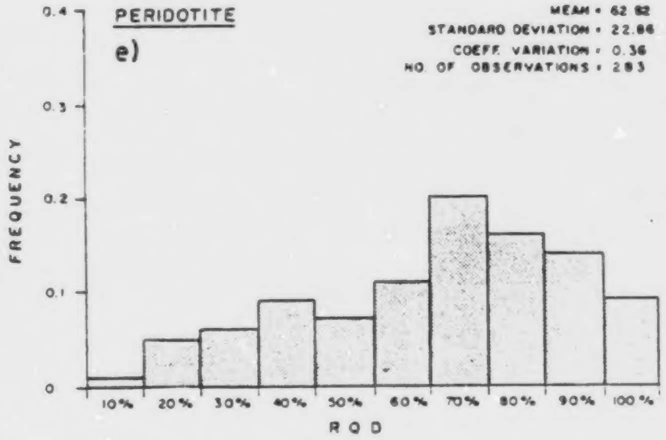
Figure **6.16**

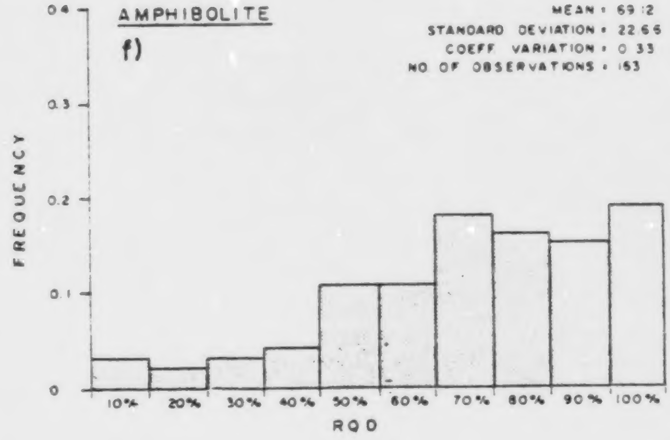


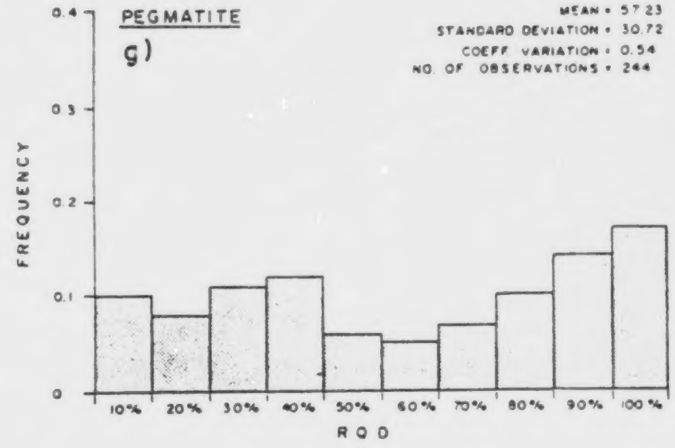
PROJECT NO. .......... DRAWN .......... REVIEWED .......... DATE ..........

# HISTOGRAMS OF ROCK QUALITY DESIGNATION HANGINGWALL ROCKS

Figure **6.16**





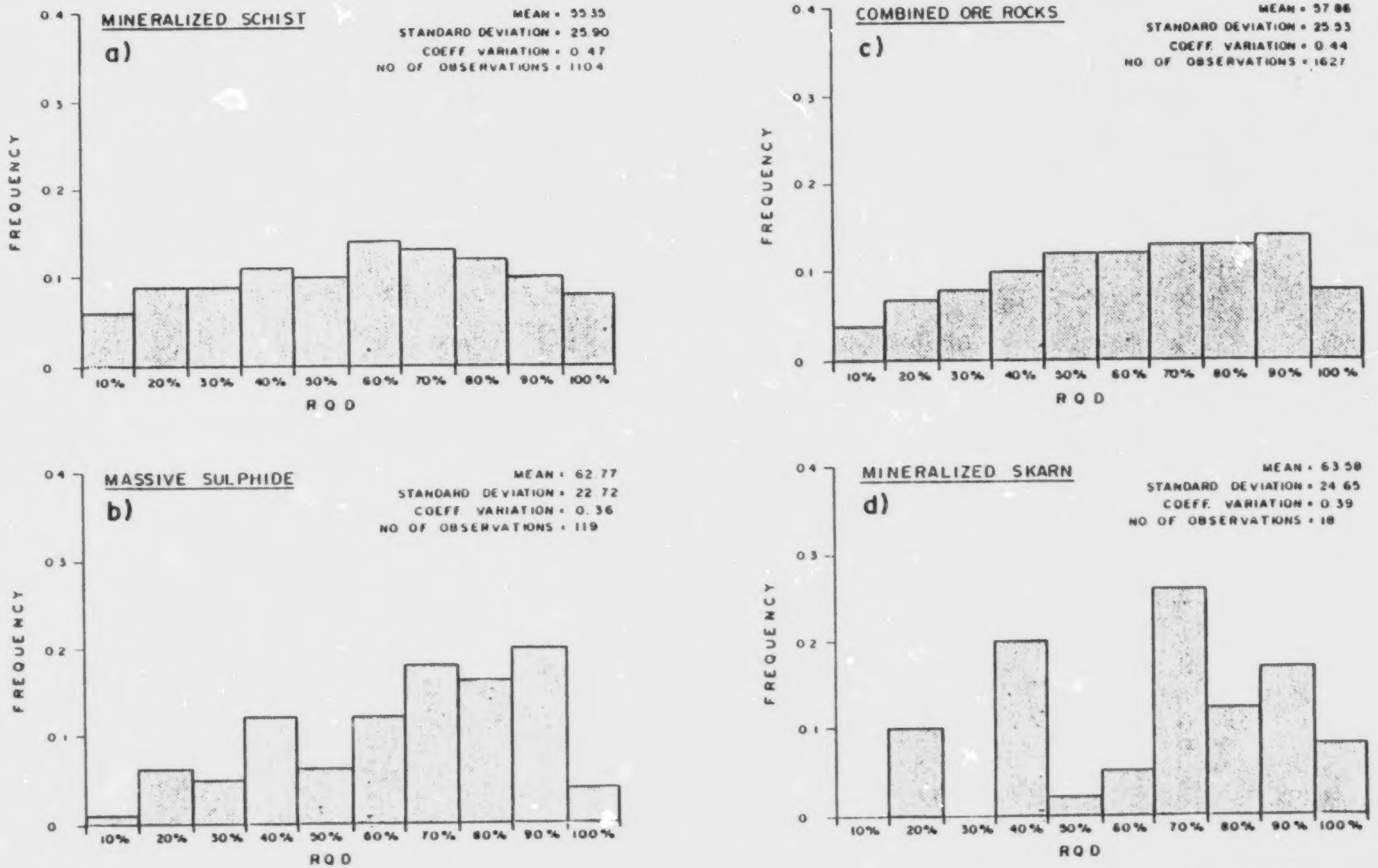


PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

# HISTOGRAMS OF ROCK QUALITY DESIGNATION ORE ROCKS

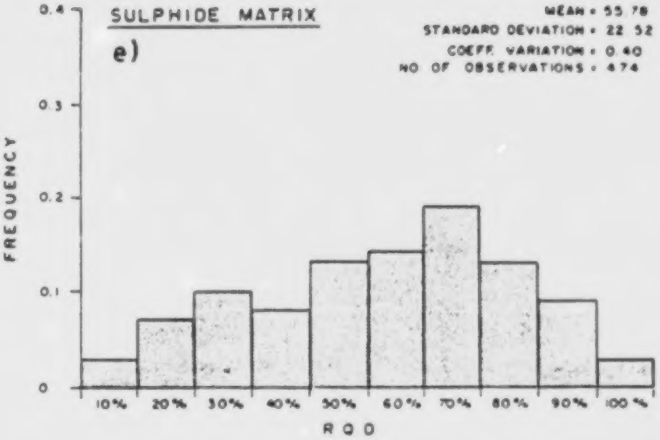
Figure **6.17**

PROJECT NO. .................... DRAWN ......... REVIEWED ......... DATE ....................



# HISTOGRAMS OF ROCK QUALITY DESIGNATION
# ORE ROCKS

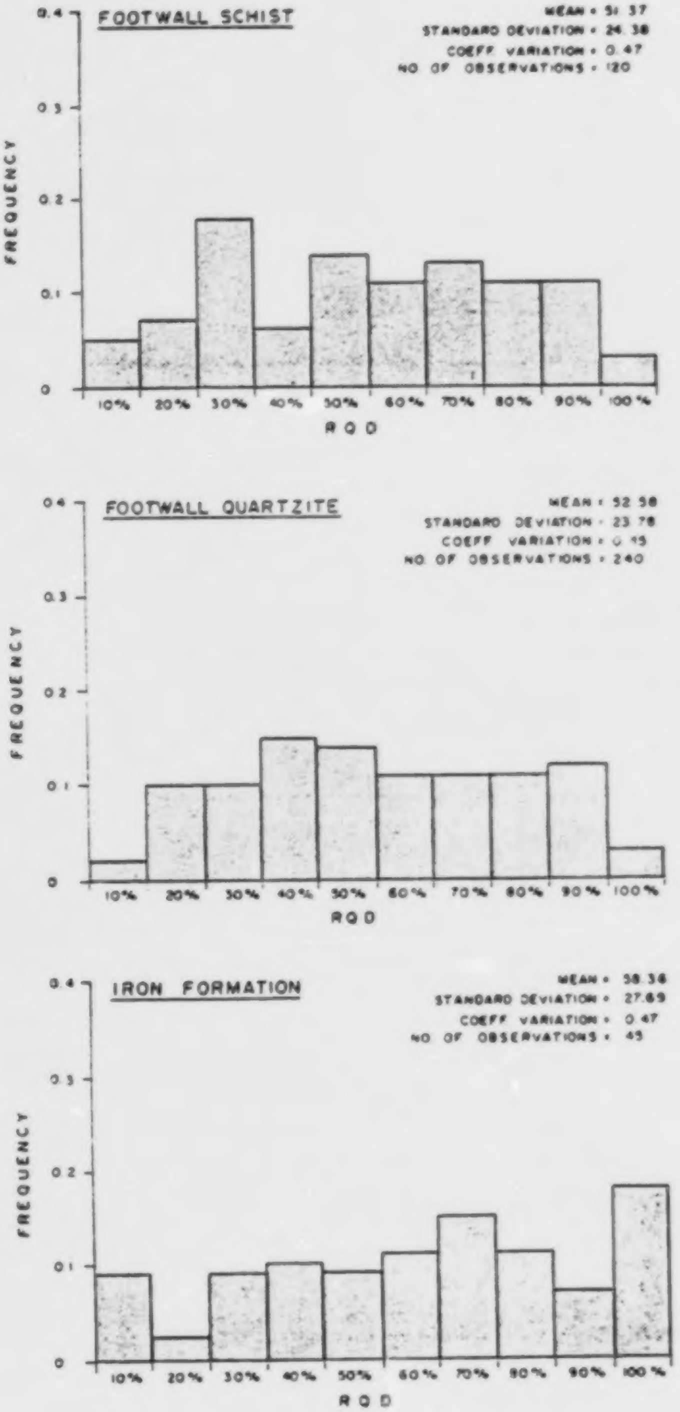
Figure **6.17**



PROJECT NO. .......... DRAWN .......... REVIEWED .......... DATE ..........

# HISTOGRAMS OF ROCK QUALITY DESIGNATION FOOTWALL ROCKS

Figure **6.18**



PROJECT NO. ..................... DRAWN ......... REVIEWED ......... DATE .....................

## List of Tables

| Table No. | DESCRIPTION |
|---|---|
| 2.1 | Prediction Process Flow Chart |
| 3.1 | Rock Strength Description |
| 3.2 | Rock Mass Classification Systems - Applications |
| 3.3 | Bieniawski RMR and Barton's Q Rock Mass Classification Systems |
| 3.4 | Number of Joints versus RQD |
| 3.5 | Joint Set Number 1 ($J_n$) |
| 3.6 | Joint Roughness Number ($J_r$) |
| 3.7 | Joint Alteration Number ($J_a$) |
| 3.8 | Joint Water Number ($J_w$) |
| 3.9 | Stress Reduction Factor, SRF |
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| 5.3 | Support Type |
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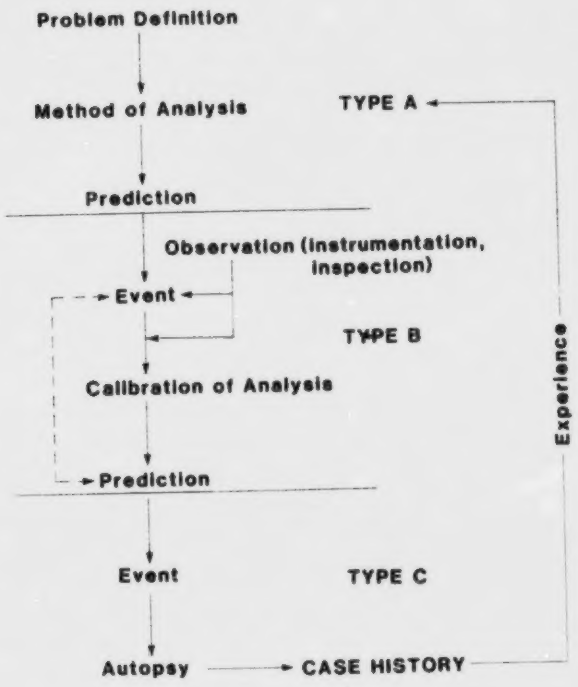
| 6.1 | Summary of Joint Orientation |
| 6.2 | Estimate of Mean Discontinuity Length |
| 6.3 | Summary of Rock Quality Data |
| 6.4 | Estimate of Mean Joint Spacing |
| 6.5 | Spacing of Joints with a Set |
| 6.6 | Rock Mass Quality |

## TABLE 2.1

## PREDICTION PROCESS FLOW CHART


Problem Definition
Method of Analysis
TYPE A
Prediction
Observation (instrumentation, inspection)
Event
TYPE B
Calibration of Analysis
Experience
Prediction
Event
TYPE C
Autopsy
CASE HISTORY

## TABLE 3.1

## ROCK STRENGTH DESCRIPTION

| Golder Assoc. Hardness Code | Geological Society Of London Term | Approx. Uniaxial Compressive Strength MPa | Approx. Uniaxial Compressive Strength psi | Field Estimation of Hardness* |
|---|---|---|---|---|
| R5 | Extremely strong | > 200 | > 29,200 | Requires many blows of geological hammer to break. |
| R4 | Very strong | 100 - 200 | 14,600 - 29,200 | Requires a few blows of geological hammer to break. |
| R3 | Strong | 50 - 100 | 7,300 - 14,600 | Breaks under single blow of geological hammer. |
| R2 | Moderately strong | 12.5 - 50 | 1,825 - 7,300 | ¼" indentations with sharp end of geological pick. |
|  | Moderately weak | 5.0 - 12.5 | 730 - 1,825 | Too hard to cut by hand into tri-axial specimen. |
| R1 | Weak | 1.25 - 5.0 | 182 - 730 | Crumbles under firm blows of geological pick. |
| S5 | Very weak (rock) | 0.60 - 1.25 | 85 - 182 | May be broken in the hand with difficulty. |
| S4 | Very stiff (soil) | 0.30 - 0.60 | 42 - 85 | Indented by finger nail |
| S3 | Stiff | 0.15 - 0.30 | 21 - 42 | Cannot be moulded in fingers. |
| S2 | Firm | 0.08 - 0.15 | 10.5 - 21 | Moulded with strong pressure of fingers. |
| S1 | Soft | 0.04 - 0.08 | 5.25 - 10.5 | Easily moulded with fingers. |

* All field estimates based on a hand held specimen of rock or rock core.

# TABLE 3.2

## ROCK MASS CLASSIFICATION SYSTEMS - APPLICATIONS
(from Pakalnis, 1986)

| | CIVIL APPLICATIONS | | | | | | | GENERAL STRENGTH INDICES | | | | MINING APPLICATIONS | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Railway Highway Dam | | | Tunnel Hydro Project | | | | | | | | Open Pit | | Underground | | | | | | | | | | |
| | Foundation Design | Slope Design | Rippability of Overburden | Excavation Method | Stand-up Time | Max. Unsupported Span | Support Design | Cohesion | Friction Angle | In-Situ Modulus of Deform. | Strength of Rock Mass | Slope Design | Rippability of Overburden | Mining Method Selection | Sequence of Mining | Stand-up Time | Max. Unsupported Span | Support Design | Pillar Design | Cavability | Extension of Caving Zone | Angle of Caving | Roof Stability | Floor Stability |
| DEERE | | | | X | | | X | | | X | | | | | | | | | | | | | | |
| BIENIAWSKI CSIR | X | X | X | X | X | X | X | X | X | X | X | X | X | | | X | X | X | X | | | | X | |
| LAUBSCHER CSIR MODIFIED | | X | | | | | X | | | | X | X | | X | X | | ? | X | X | X | X | X | X | |
| BARTON NGI | | | | | X | X | X | X | X | X | X | | | | | X | X | X | X | | | | X | |
| KIRSTEN, NGI MODIFIED | | | X | | | | X | | | | | | X | | | | | X | | | | | X | |
| WICKHAM | | | | | | | X | | | | | | | | | | | | | | | | | |
| STINNI LAUFFER | | | | | X | X | | | | | | | | | | | | | | | | | | |
| TERZAGHI | | | | | | | X | | | | | | | | | | | | | | | | | |
| FRANKLIN | | | | | X | X | | | | | | | | | | | | | | | | | | |
| MATHEWS | | | | | | | | | | | | | | | | | X | | ? | | | | X | X |

## TABLE 3.3

## ROCK MASS CLASSIFICATION

### 1. BIENIAWSKI RMR ROCK MASS CLASSIFICATION SYSTEM

**RMR = A + B + C + D + E - F**

**A = rating for uniaxial compressive strength**

**B = rating for Rock Quality Designation**

**C = rating for joint spacing**

**D = rating for joint condition**

**E = rating for ground water conditions**

**F = adjustment for joint orientation**

### 2. BARTON'S Q ROCK MASS CLASSIFICATION SYSTEM

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

| | | |
|---|---|---|
| RQD | = Rock Quality Designation | Block Size |
| $J_n$ | = Joint Set Number | |
| $J_r$ | = Joint Roughness Number | Shear Strength |
| $J_a$ | = Joint Alteration Number | |
| $J_w$ | = Joint Water Reduction | Active Stress |
| SRF | = Stress Reduction Factor | |

## TABLE 3.4

## NUMBER OF JOINTS VERSUS RQD

| No. of Joints Per Cu. Metre (Jc) | Ground Quality Designation (RQD) | No. of Joints per Cu. Metre (Jc) | Ground Quality Designation (RQD) |
|---|---|---|---|
| 33 | 5 | 18 | 55 |
| 32 | 10 | 17 | 60 |
| 30 | 15 | 15 | 65 |
| 29 | 20 | 14 | 70 |
| 27 | 25 | 12 | 75 |
| 26 | 30 | 11 | 80 |
| 24 | 35 | 9 | 85 |
| 23 | 40 | 8 | 90 |
| 21 | 5 | 6 | 95 |
| 20 | 50 | 5 | 100 |

Alternatively RQD may be estimated from mean discontinuity spacing with the aid of Figure 3.22.

## TABLE 3.5

## JOINT SET NUMBER 1 (Jn)

| Number of Joint Sets | Joint Set Number (Jn) |
|---|---|
| Intact, no or few joint/fissures | 0.5 - 1.0 |
| One joint/fissure set | 2 |
| One joint/fissure set plus random | 3 |
| Two joint/fissure sets | 4 |
| Two joint/fissure sets plus random | 6 |
| Three joint/fissure sets | 9 |
| Three joint/fissure sets plus random | 12 |
| Four or more joint/fissure sets, random, heavily jointed, sugar cube, etc. | 15 |
| Multiple joint/fissure sets (crushed rock, earth-like) | 20 |

## TABLE 3.6

## JOINT ROUGHNESS NUMBER (Jr)

| Joint Separation | Condition of Joint | Joint Roughness Number (Jr) |
|---|---|---|
| Joints/Fissures Tight or Losing during Excavation | Discontinuous Joint/ Fissures | 4.0 |
| | Rough or Irregular, Indulating | 3.0 |
| | Smooth Indulating | 2.0 |
| | Rough or Irregular, Planar | 1.5 |
| | Smooth Planar | 1.0 |
| | Slickensided Planar | 0.5 |
| Joints/Fissures Open and Remain Open during Excavation | Joints/Fissures either Open or Containing Relatively Soft Gouge or Sufficient Thickness to Prevent Joint/Fissure Wall Contact upon Excavation | 1.0 |
| | Shattered or Micro-Shattered Clays | 1.0 |

## TABLE 3.7

## JOINT ALTERATION NUMBER, Ja

| TYPICAL DESCRIPTION | VALUE |
|---|---|
| Tightly Healed | 0.75 |
| Surface Staining Only | 1.0 |
| Low Friction Thin Coating (chlorite, talc, clay, etc.) | 4.0 |
| Thin Swelling Clay | 10.0 |
| Thick Swelling Clay | 20.0 |

NOTES:
1. Joint walls come into contact before 100 mm shear.
2. Joint walls do not come into contract at all upon shear.

## TABLE 3.8

## JOINT WATER NUMBER (Jw)

| Condition of Groundwater | Head of Water (m) | Joint Water Reduction Factor (Jw) |
|---|---|---|
| Dry Excavation or Minor Inflow 5 litre/minute locally | <10 | 1.0 |
| Medium Inflow, Occasional Outwash of Joint/Fissure Fillings | 10 - 25 | 0.66 |
| Large Inflow in Competent Ground with Unfilled Joints/ Fissures | 25 - 100 | 0.5 |
| Large Inflow with Considerable Outwash of Joint/Fissure Fillings | 25 - 100 | 0.33 |
| Exceptionally High Flow upon Excavation, Decaying with Time | >100 | 0.2 - 0.1 |
| Exceptionally High Inflow Continuing without Noticeable Decay | >100 | 0.1 - 0.05 |

NOTES:

1. Last three categories are crude estimates. Increase $J_w$ if drainage measures are installed.
2. Special problems caused by ice formation are not considered.

## TABLE 3.9

## STRESS REDUCTION FACTOR, SRF

(a) Weakness zones intersecting excavation which may cause loosening of rock mass when opening is excavated.

| Description | SRF Value |
|---|---|
| Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock (any depth) | 10 |
| Single weakness zones containing clay or chemically disintegrated rock (depth of excavation <50m) | 5 |
| Single weakness zones containing clay or chemically disintegrated rock (depth of excavation >50m) | 2.5 |
| Multiple shear zones in competent rock (clay-free), loose surrounding rock (any depth) | 7.5 |
| Single shear zones in competent rock (clay-free) (depth of excavation <50m) | 5.0 |
| Single shear zones in competent rock (clay-free) (depth of excavation >50m) | 2.5 |
| Loose open joints, heavily jointed or "sugar-cube" etc. (any depth) | 5.0 |

NOTES:

1. Reduce these values of SRF by 25-50% if the relevant shear zones only influence, but do not intersect the excavation.

## TABLE 3.9 - STRESS REDUCTION FACTOR, SRF Cont'd

(b) Competent rock, rock stress problems

| Description | $\sigma_c/\sigma_1$ | $\sigma_t/\sigma_1$ | SRF Value |
|---|---|---|---|
| Low Stress, Near-Surface | >200 | >13 | 2.5 |
| Medium Stress | 200-10 | 13-0.66 | 1.0 |
| High Stress, Very Tight Structure (Usually Favourable For Wall Stability) | 10-5 | 0.66-0.33 | 0.5-2 |
| Mild Rock Burst Massive Rock) | 5-2.5 | 0.33-0.16 | 5-10 |
| Heavy Rock Burst (Massive Rock) | <2.5 | <0.16 | 10-20 |

## TABLE 3.10

## GEOMECHANICS CLASSIFICATION OF JOINTED ROCK MASSES

*(a) Classification parameters and their ratings*

| Parameter | | Ranges of values | | | | | | |
|---|---|---|---|---|---|---|---|---|
| 1 strength of intact rock material | point-load strength index (MPa) | >8 | 4–8 | 2–4 | 1–2 | for this low range, uniaxial compression test is preferred | | |
| | uniaxial compressive strength (MPa) | >200 | 100–200 | 50–100 | 25–50 | 10–25 | 3–10 | 1–3 |
| rating | | 15 | 12 | 7 | 4 | 2 | 1 | 0 |
| 2 drill core quality *RQD* (%) | | 90–100 | 75–90 | 50–75 | 25–50 | <25 | | |
| rating | | 20 | 17 | 13 | 8 | 3 | | |
| 3 joint spacing (m) | | >3 | 1–3 | 0.3–1 | 0.05–0.3 | <0.05 | | |
| rating | | 30 | 25 | 20 | 10 | 5 | | |
| 4 condition of joints | | very rough surfaces, not continuous, no separation, hard joint wall rock | slightly rough surfaces, separation <1 mm, hard joint wall rock | slightly rough surfaces, separation <1 mm, soft joint wall rock | slickensided surfaces<br>or<br>gouge <5 mm thick<br>or<br>joints open 1–5 mm, continuous joints | soft gouge > 5 mm thick<br>or<br>joints open >5 mm, continuous joints | | |
| rating | | 25 | 20 | 12 | 6 | 0 | | |
| 5 groundwater | inflow per 10 m tunnel length ($\ell\ \mathrm{min}^{-1}$) | none | | <25 | 25–125 | >125 | | |
| | or<br>$\frac{\text{joint water pressure}}{\text{major principal stress}}$ | 0 | | 0.0–0.2 | 0.2–0.5 | >0.5 | | |
| | or<br>general conditions | completely dry | | moist only (interstitial water) | water under moderate pressure | severe water problems | | |
| rating | | 10 | | 7 | 4 | 0 | | |

*(b) Rating adjustment for joint orientations*

| Strike and dip orientations of joints | | very favourable | favourable | fair | unfavourable | very unfavourable |
|---|---|---|---|---|---|---|
| Ratings | tunnels | 0 | −2 | −5 | −10 | −12 |
| | foundations | 0 | −2 | −7 | −15 | −25 |
| | slopes | 0 | −5 | −25 | −50 | −60 |

*(c) Rock mass classes determined from total ratings*

| Ratings | 100 ← 81 | 80 ← 61 | 60 ← 41 | 40 ← 21 | <20 |
|---|---|---|---|---|---|
| Class no. | I | II | III | IV | V |
| Description | very good rock | good rock | fair rock | poor rock | very poor rock |

*(d) Meaning of rock classes*

| Class no. | I | II | III | IV | V |
|---|---|---|---|---|---|
| average stand-up time | 10 years for 5 m span | 6 months for 4 m span | 1 week for 3 m span | 5 hours for 1.5 m span | 10 minutes for 0.5 m span |
| cohesion of the rock mass (kPa) | >300 | 200–300 | 150–200 | 100–150 | <100 |
| friction angle of the rock mass | >45° | 40°–45° | 35°–40° | 30°–35° | <30° |

**(after BIENIAWSKI 1976)**

## TABLE 4.1

## DESCRIPTION OF ROCK MASSES

### PARTICLE SIZE, DISCONTINUITY & BEDDING SPACING

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| DISCONTINUITY SPACING DESCRIPTIONS | | 20mm | 60mm | 200mm | 600mm | 2000mm | 6000mm |
| | EXTREMELY CLOSE SPACING | VERY CLOSE SPACING | CLOSE SPACING | MODERATE SPACING | WIDE SPACING | VERY WIDE SPACING | EXTREMELY WIDE SPACING |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| BEDDING | FOLIATED OR CLEAVED | VERY INTENSELY LAMINATED | INTENSELY LAMINATED | VERY THINLY | THINLY | MEDIUM | THICKLY | VERY THICKLY |
| | | | | 20mm | 60mm | 200mm | 600mm | 2000mm |

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| PARTICLE SIZE DESCRIPTIONS | CLAY | SILT | | | SAND | | | GRAVEL | | | BOULDERS |
| | | FINE | MEDIUM | COARSE | FINE | MEDIUM | COARSE | FINE | MEDIUM | COARSE | |
| | | 0.002mm | 0.006mm | 0.02mm | 0.06mm | 0.2mm | 0.6mm | 2mm | 6mm | 20mm | 60mm 200mm 600mm 2000mm |

| | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| STRENGTH DESCRIPTIONS OF CLAYS AND ROCKS | V. SOFT CLAY | SOFT CLAY | FIRM CLAY | STIFF CLAY | V. STIFF CLAY | HARD CLAY | EXTREMELY WEAK ROCK | V. WEAK ROCK | WEAK ROCK | MED. STRONG ROCK | STRONG ROCK | V. STRONG ROCK | EXT. STRONG ROCK |
| | S1 | S2 | S3 | S4 | S5 | S6 | R0 | R1 | R2 | R3 | R4 | R5 | R6 |
| | | 0.025 MPa | 0.05 MPa | 0.10 MPa | 0.25 MPa | 0.5 MPa | | 1.0 MPa | 5.0 MPa | 25 MPa | 50 MPa | 100 MPa | 250 MPa |

| | | | | | |
|---|---|---|---|---|---|
| PERSISTENCE | | 1m | 3m | 10m | 20m |
| | VERY LOW PERSISTENCE | LOW PERSISTENCE | MEDIUM PERSISTENCE | HIGH PERSISTENCE | VERY HIGH PERSISTENCE |

| | | | | | | |
|---|---|---|---|---|---|---|
| SEPARATION OF FRACTURE WALLS | | 0mm | 0.1mm | 1.0mm | 5.0mm | 25mm |
| | CLOSED | VERY NARROW | NARROW | WIDE | VERY WIDE | |

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| ROUGHNESS CLASSIFICATION | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 |
| | ROUGH STEPPED | SMOOTH STEPPED | SLICKENSIDED | ROUGH UNDULATING | SMOOTH UNDULATING | SLICKENSIDED UNDULATING | ROUGH PLANAR | SMOOTH PLANAR | SLICKENSIDED PLANAR |

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| APERTURE | 'CLOSED' FEATURES | | | 'GAPPED' FEATURES | | | 'OPEN' FEATURES | | |
| | VERY TIGHT | TIGHT | PARTLY OPEN | OPEN | MODERATELY WIDE | WIDE | VERY WIDE | EXTREMELY WIDE | CAVERNOUS |
| | | 0.1 mm | 0.25mm | 0.5 mm | 2.5mm | 10mm | 1cm | 10cm | 100cm |

| | | | | | | |
|---|---|---|---|---|---|---|
| BLOCK SIZE | MASSIVE | BLOCKY | TABULAR | COLUMNAR | IRREGULAR | CRUSHED |
| | FEW JOINTS OR VERY WIDE SPACING | APPROXIMATELY EQUIDIMENSIONAL | ONE DIMENSION CONSIDERABLY SMALLER THAN THE OTHER TWO | ONE DIMENSION CONSIDERABLY LARGER THAN THE OTHER TWO | WIDE VARIATIONS OF BLOCK SIZE AND SHAPE | HEAVILY JOINTED TO 'SUGAR CUBE' |

| | | | | | | |
|---|---|---|---|---|---|---|
| SEEPAGE | I | II | III | IV | V | VI |
| | DISCONTINUITY IS DRY, WATER FLOW DOES NOT APPEAR POSSIBLE | DISCONTINUITY IS DRY OR DAMP, BUT THERE IS NO FREE WATER OR FLOW | DISCONTINUITY IS DAMP OR WET AND INDICATES SIGNS OF WATER FLOW | DISCONTINUITY SHOWS SIGNS OF CONTINUOUS WATER FLOW (ESTIMATE l MIN) | DISCONTINUITY SHOWS SEEPAGE AND DROPS OF WATER (ESTIMATE l MIN) | DISCONTINUITY SHOWS CONTINUOUS FLOW OF WATER (ESTIMATE l/MIN & DESCRIBE PRESSURE |

### DEGREES OF WEATHERING

| DIAGNOSTIC FEATURE<br>DESCRIPTION | DISCOLORATION EXTENT | FRACTURE CONDITION | SURFACE CHARACTERISTICS |
|---|---|---|---|
| UNWEATHERED | NONE | CLOSED OR DISCOLORED | UNCHANGED |
| SLIGHTLY WEATHERED | <20% OF FRACTURE SPACING ON BOTH SIDES OF FRACTURE | DISCOLORED MAY CONTAIN THIN FILLING | PARTIAL DISCOLORATION |
| MEDIUM WEATHERED | >20% OF FRACTURE SPACING ON BOTH SIDES OF FRACTURE | DISCOLORED MAY CONTAIN THICK FILLING | PARTIAL TO COMPLETE DISCOLORATION NOT FRIABLE EXCEPT POORLY CEMENTED ROCKS |
| HIGHLY WEATHERED | THROUGHOUT | | FRIABLE AND POSSIBLY PITTED |
| COMPLETELY WEATHERED | THROUGHOUT | | RESEMBLES SOIL |

### CLASSIFICATION OF ROCK HARDNESS

| DESCRIPTION | FIELD IDENTIFICATION |
|---|---|
| R0 EXTREMELY WEAK ROCK | INDENTED BY THUMBNAIL |
| R1 VERY WEAK ROCK | CRUMBLES UNDER FIRM BLOWS OF GEOLOGICAL HAMMER CAN BE PEELED BY A POCKET KNIFE |
| R2 WEAK ROCK | CAN BE PEELED BY A POCKET KNIFE WITH DIFFICULTY. SHALLOW INDENTATIONS MADE BY FIRM BLOWS OF GEOLOGICAL PICK |
| R3 MEDIUM STRONG ROCK | CANNOT BE SCRAPPED OR PEELED WITH KNIFE. CAN BE FRACTURED WITH SINGLE FIRM BLOW WITH GEOLOGICAL HAMMER |
| R4 STRONG ROCK | SPECIMEN REQUIRES MORE THAN ONE BLOW WITH GEOLOGICAL HAMMER TO FRACTURE IT |
| R5 VERY STRONG ROCK | SPECIMEN REQUIRES MANY BLOWS TO FRACTURE IT |
| R6 EXTREMELY STRONG ROCK | SPECIMEN CAN ONLY BE CHIPPED WITH GEOLOGICAL HAMMER |

### ROCK CORE SIZES

| CORE BARREL INDEX LETTER | CORE (mm) DIAMETER |
|---|---|
| EX | 20.6 |
| AX | 30.2 |
| BX | 41.3 |
| NX | 54.0 |
| HX | 76.2 |
| PF | 92.1 |
| SF | 112.7 |
| UF | 139.7 |
| ZF | 165.1 |

### ROCK CORE DESCRIPTIONS

| | |
|---|---|
| TCR | TOTAL CORE RECOVERY (THE LENCTH OF THE TOTAL AMOUNT OF CORE SAMPLE RECOVERED EXPRESSED AS A PERCENTAGE OF THE LENGTH OF CORE RUN) |
| SCR | SOLID CORE RECOVERY (THE LENGTH OF CORE RECOVERED AS SOLID CYLINDERS, EXPRESSED AS A PERCENTAGE OF THE LENGTH OF CORE RUN) |
| ROD | ROCK QUALITY DESIGNATION (THE LENGTH OF CORE RECOVERED AS SOLID CYLINDERS, EACH MORE THAN 100mm LONG EXPRESSED AS A PERCENTAGE OF THE LENGTH OF CORE RUN) |

## TABLE 5.1

## STRESS STATE

| Condition | Description |
|---|---|
| No Damage Evident | Walls of area generally sound, structurally controlled failure only. |
| Slight Stress Damage | Some slabbing of walls of area not evidently controlled by structure; drill holes non-circular after a period of time; give direction of long axis. |
| High Stress Damage | Area has blasted appearance, some difficulty in withdrawing drill steel. Dogearing of drill holes. |
| Failed | Area generally heavily fractured and of a loose nature (slack), considerable difficulty in drilling due to shearing and sloughing in drill holes. |

## TABLE 5.2

## SCALING EFFORT

| Condition | Description |
|---|---|
| Minor Scaling | General scaling of back and walls to remove small loose (<1 ft. maximum dimension) |
| Moderate Scaling | Several large blocks of loose (1 ft to 5 ft maximum dimension) exists |
| Heavy Scaling | Numerous large blocks of loose exist. Ground working need constant rescaling in order to secure |

## TABLE 5.3

## SUPPORT TYPE

| Category | Description |
|---|---|
| 0 | None |
| 1 | Spot bolting |
| 2 | Pattern bolting - mechanical bolt |
| 3 | Pattern bolting - friction bolt |
| 4 | Pattern bolting - cable bolt |
| 5 | Pattern bolting plus shotcrete |
| 6 | Timber sets or cribs |
| NOTE: A postscript of M or S to the category number will indicate the addition of mesh or straps respectively | |

## TABLE 5.4

## SUPPORT LOAD

| Category | Description |
|---|---|
| 0 | No visible signs of load |
| B1 | Bolt washer convex |
| B2 | Bolt washer flat |
| B3 | Bolter washer concave |
| B4 | Bolt failed |
| B5 | Bolt loose in hole |
| T1 | Post and beam intact, wedges tight |
| T2 | Post splitting |
| T3 | Beams splitting |
| T4 | Post and beam splitting |

## TABLE 6.1

## SUMMARY OF JOINT ORIENTATION

| JOINT SET NO. | DIP DIRECTION (°)* | | DIP ANGLE (°) | |
|---|---|---|---|---|
| | MODE | RANGE | MODE | RANGE |
| 1 | 100 | 76-124 | 70 | 45-85 |
| 2 | 167/347 | 154-197<br>334-017 | 90 | 64-90 |
| 3 | 247/067 | 243-250<br>063-070 | 90 | 75-90 |
| 4 | 310 | 275-360 | 20 | 0-40 |

*NOTE: DIP DIRECTION W.R.T. MINE NORTH

## TABLE 6.2

## ESTIMATE OF MEAN DISCONTINUITY LENGTH

| Discontinuity Class | Estimated Mean Length (m) |
|---|---|
| Foliation, (Joint Set 1), Minor Faults and Shear Zones | 50 |
| Joints Parallel MacKenzie Dyke Swarm (Joint Set 2) | 5 |
| Joints Perpendicular MacKenzie Dyke Swarm (Joint Set 3) | 5 |
| Joints Perpendicular Foliation (Joint Set 4) | 2 |

## TABLE 6.3

## SUMMARY OF ROCK QUALITY DATA

| UNIT | ROCK TYPE | ROCK QUALITY DESIGNATION | | | | |
|---|---|---|---|---|---|---|
| | | MEAN (%) | MODE(%) | COEFF. Var.(%) | RANGE (%) | NO. OBS. |
| Hangingwall | Hangingwall Schist | 61 | 80 | 44 | 0-100 | 478 |
| | Hangingwall Quartzite | 58 | 70 | 44 | 0-100 | 455 |
| | Skarn | 70 | 80 | 31 | 30-100 | 209 |
| | Gneiss | 75 | 100 | 39 | 60-100 | 277 |
| | Peridotite | 63 | 70 | 36 | 10-100 | 283 |
| | Amphibolite | 69 | 70 | 33 | 10-100 | 163 |
| | Pegmatite | 57 | 100 | 54 | 0-100 | 244 |
| Ore | Skarn | 64 | 70 | 39 | 10-100 | 718 |
| | Schist | 55 | 60 | 47 | 0-100 | 1104 |
| | Massive Sulphide | 63 | 90 | 36 | 20-100 | 119 |
| | Sulphide Matrix | 56 | 70 | 40 | 10-100 | 474 |
| Footwall | Schist | 51 | 30 | 47 | 0-100 | 120 |
| | Quartzite | 53 | 45 | 45 | 0-100 | 240 |
| | Iron Formation | 58 | 100 | 47 | 0-100 | 45 |

## TABLE 6.4

## ESTIMATE OF MEAN JOINT SPACING

| STRUCTURE | ROCK TYPE | MEAN RQD (%) | EST. MEAN JOINT SPACING(m) |
|---|---|---|---|
| Hanging wall | Quartzite | 58 | 0.07 |
| | Biotite Gneiss | 100 | 0.35 |
| | Skarn | 70 | 0.09 |
| | Schist | 61 | 0.07 |
| | Amphibolite | 69 | 0.09 |
| | Peridotite | 63 | 0.08 |
| | Pegmatite | 57 | 0.07 |
| Ore | Massive Sulphide | 63 | 0.08 |
| | Brecciated Sulphide | 56 | 0.07 |
| | Mineralized Schist | 55 | 0.07 |
| | Mineralized Skarn | 64 | 0.08 |
| Footwall | Schist | 51 | 0.06 |
| | Quartzite | 52 | 0.06 |
| | Iron Formation | 58 | 0.07 |

## TABLE 6.5

## SPACING OF JOINTS WITH A SET

| ROCK TYPE | EST. MEAN SPACING IN JOINT SET (m) 1 | 2 | 3 | 4 |
|---|---|---|---|---|
| Quartzite | 0.4 | 2.3 | 3.0 | 0.6 |
| Skarn | 0.35 | 4.0 | 8.0 | 0.6 |

## TABLE 6.6

## ROCK MASS QUALITY

| ROCK TYPE | | | PARAMETERS | | | | | ROCK MASS QUALITY (Q) | |
|---|---|---|---|---|---|---|---|---|---|
| | RQD | | | | | | | | |
| | MEAN | RANGE | Jn | Jr | Ja | Jw | SRF | MEAN | RANGE |
| Biotite Gneiss | 100 | 60-100 | 9 | 1.5 | 0.75 | 1 | 2 | 11 | 7-20 |
| Amphibolite | 70 | 10-100 | 9 | 1.5 | 0.75 | 1 | 2 | 8 | 1-11 |
| Skarn (unleached) | 80 | 30-100 | 9 | 1.5 | 0.75 | 1 | 2 | 9 | 3-11 |
| Skarn (leached) | 10 | --- | 12 | 1.0 | 5.0 | 1 | 5 | 0.03 | --- |
| H.W. Quartzite | 70 | 10-100 | 9 | 1.5 | 0.75 | 1 | 2 | 8 | 1-11 |
| H.W. Schist | 65 | 10-100 | 9 | 2.0 | 2.0 | 1 | 2 | 4 | 0.6-6 |
| Ore Rocks | 55 | 10-100 | 9 | 1.5 | 1.0 | 1 | 2 | 5 | 0.8-8 |
| F.W. Schist | 30 | 10-100 | 9 | 1.5 | 2.0 | 1 | 2 | 1 | 0.4-4 |
| F.W. Quartzite | 50 | 10-100 | 9 | 2.0 | 0.75 | 1 | 2 | 7 | 1-15 |

# Volume

# 2

# MINERALS

## ECONOMIC & REGIONAL DEVELOPMENT AGREEMENT

Canada

Manitoba

MG-13097A

# CANADA - MANITOBA
# MINERAL DEVELOPMENT AGREEMENT

## RESEARCH AND DEVELOPMENT IN MANITOBA MINES

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## TASK 4 - GEOTECHNICAL DATA BASE

## FOR GROUND CONTROL

Geomechanical Modelling
Final Report - 1989
Volume II - Appendices

by

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# APPENDIX A

## Standard Procedure for Collection of Rock Mass Classification Data from Underground Exposures

| | |
|---|---|
| A1 to A3 | Standard Procedure for Collection of Rock Mass Classification Data for Underground Exposures |
| A4 to A5 | Data Collection Sheet |

# APPENDIX A

## Standard Procedure for Collection of Rock Mass Classification Data from Underground Exposures

The proposed data collection form is given in Figure A1. The following procedure is recommended for carrying out rock mass classification measurements along an underground drift exposure:

1) Select area to be classified; typically this will be the wall or back of a haulage, drill drift or drawpoint. The sampling site should cover an area of approximately $30m^2$ and should consist of predominantly one rock type.

2) Location: record location and date of observation.

3) Strength: The strength (or hardness) of the rock should be recorded using the following field method: -

| Grade | Description | Field Identification | Approx. Range of Uniaxial Compressive Strength (MPa) |
|---|---|---|---|
| S1 | Very Soft Clay | Easily Penetrated Several Inches by fist | <0.025 |
| S2 | Soft Clay | Easily Penetrated Several Inches by Thumb | 0.025-0.05 |
| S3 | Firm Clay | Can be Penetrated Several Inches by Thumb with Moderate Effort | 0.05-0.10 |
| S4 | Stiff Clay | Readily Indented by Thumb but Penetrated only with Great Effort | 0.10-0.25 |
| S5 | Very Stiff Clay | Readily Indented by Thumbnail | 0.25-0.50 |
| S6 | Hard Clay | Indented with Difficulty by Thumbnail | >0.50 |
| RO | Extremely Weak Rock | Indented by Thumbnail | 0.25-1.0 |
| R1 | Weak Rock | Crumbles under Firm Blows with Point of Geological Hammer, can be Peeled by a Pocket Knife | 1.0-5.0 |
| R2 | Weak Rock | Can be Peeled by a Pocket Knife with Difficulty, Shallow indentations made by Firm Blow with Point of Geological Hammer | 5.0-2.5 |

| Grade | Description | Field Identification | Approx. Range of Uniaxial Compressive Strength (MPa) |
|---|---|---|---|
| R3 | Medium Strong Rock | Cannot be Scraped or Peeled with a Pocket Knife, Specimen can be Fractured with Single Firm Blow of Geological Hammer | 25-50 |
| R4 | Strong Rock | Specimen Requires more one Blow of Geological Hammer to Fracture it | 50-100 |
| R5 | Very Strong Rock | Specimen Requires many Blows of Geological Hammer to Fracture it | 100-250 |
| R6 | Extremely Strong Rock | Specimen can only be Chipped with Geological Hammer | >250 |

4) Joint Set Number: From inspection determine the number of joint sets present (using the Table on the data collection sheet as a guide). If joint set orientations are reasonably consistent, as typically has been found to be the case for orebodies in foliated or bedded rocks, this is a straightforward task. Visual recognition of the number of sets should be accompanied by some system of numbering for identification purposes. For example, the most systematic and persistent set can be labelled "set No. 1" and so on.

Notes: Systematic joint sets should be distinguished from non-systematic joints when recording the number of sets. In general systematic joints will be persistent features, with individual joints parallel or sub-parallel in plan, while non-systematic joints display random rather than oriented patterns in plan and section.
Incipient discontinuities such as those that may develop parallel to bedding, or parallel to foliation or cleavage, should be included in the local estimate of the number of sets, if it is considered that the method of excavation employed will sufficiently disturb the rock mass to cause development of these features into equivalent bedding joints, foliation joints, etc.

As noted above, the number of sets recorded will tend to be a function of the size of area mapped, and should be interpreted accordingly. The spacing of individual sets will play an important role in this interpretation. For example, four sets recognized following a "conventional" survey of an area (using the pole contouring method) may include some sets with such wide spacing that these would be of little relevance to the stability of a short length of tunnel.

5) Orientation: For each of the joint sets defined measure the mean discontinuity orientation. Once familar with the geologic fabric of the area, this may be done by inspection. However, until that time occurs 3 to 5 independent measurements of dip and dip direction should be taken and averaged. Dip should be recorded, to the nearest 5°, dip direction to the nearest 10°.

6) **Spacing:** For each joint set identified, the mean discontinuity spacing or frequency should be determined. This measurement can be rapidly performed as individual joint spacings are not of interest. 5 or 10m can be paced out or estimated with reasonable accuracy by most observers (i.e. to within + or - 10% of the correct length). The observer should face in the direction of strike (if space permits) for each joint set that is to be counted and count perpendicular to the strike, thereby removing the angular correction factor. Mean spacing or discontinuity frequency over the sampled length can then be quickly determined.

7) **Block Shape:** The exposure can be described by the following adjectives to give an impression of block size and shape:

(i) massive = few joints or very wide spacing
(ii) blocky = approximately equidimensional
(iii) tabular = one dimension considerably smaller than the other two
(iv) columnar = one dimension considerably larger than the other two
(v) irregular = wide variations of block size and shape
(vi) crushed = heavily jointed to "sugar cube"

8) **Joint Surface Alteration:** For each joint set identified, the typical or average alteration condition should be recorded using the table on the field data collection sheet as a guide. This is done in two steps. First the separation between the joint surfaces is determined and secondly the appropriate infill description in the table identified.

9) **Joint Surface Roughness:** For each joint set identified, the typical or average surface roughness should be recorded using the table on the field data collection sheet as a guide.

10) **Stress Reduction Factor:** Where appropriate the stress reduction factor should be recorded using the table in the data collection sheet as a guide. If conditions observed are not found in the table SRF should be set to one.

11) **Other Information:** Other information on the form includes Q, MRMR, average volume and RQD. Values for these can be estimated from the data recorded above.

12) A typical field sheet with all data duly entered is shown in Figure A2.

| | |
|---|---|
| **DATA COLLECTION SHEET**<br>**ROCK MASS CLASSIFICATION** | Figure **A1** |

| | |
|---|---|
| LOCATION:<br>Level ____________ Face __________ Distance ________ | DATE |

| Rock Type | Strength | Sketch |
|---|---|---|
| | | |

| Joint Data | Set 1 | Set 2 | Set 3 | Set 4 |
|---|---|---|---|---|
| Dip Direct/Dip Angle | | | | |
| Average Spacing (ft) | | | | |
| Estimated No. of Joints/m$^3$ | | | | |

Q =

MRMR =

Avg. Vol. =

Shape

RQD

## JOINT ALTERNATION NUMBER (Ja)

| Description of gouge | Joint alteration number (Ja) for joint separation (mm) | | |
|---|---|---|---|
| | <1.01[1] | 1.0-5.0[2] | >5.0[3] |
| Tightly healed, hard, nonsoftening impermeable filling | 0.75 | - | - |
| Unaltered joint walls, surface staining only | 1.0 | - | - |
| Slightly altered, non-softening, non-cohesive rock mineral or crushed rock filling | 2.0 | 4.0 | 6.0 |
| Non-softening, slightly clayey non-cohesive filling | 3.0 | 6.0 | 10.0 |
| Softening or low friction clay mineral coatings and small quantities of swelling clays | 4.0 | 8.0[3] | 13.0 |
| Shattered or micro-shattered (swelling) clay gouge, with or without crushed rock | 5.0 | 10.0[4] | 18.0 |

## JOINT ROUGHNESS NUMBER (Jr)

| Joint Separation | Condition of Joint | Joint Roughness Number (Jr) |
|---|---|---|
| Joints/fissures tight or losing during excavation | Discontinuous joint/fissures | 4.0 |
| | Rough or irregular, undulating | 3.0 |
| | Smooth undulating | 2.0 |
| | Rough or irregular, planar | 1.5 |
| | Smooth planar | 1.0 |
| | Slickensided planar | 0.5 |
| Joints/fissures open and remain open during excavation | Joints/fissures either open or containing relatively soft gouge or sufficient thickness to prevent joint/fissure wall contact upon excavation. | 1.0 |
| | Shattered or micro-shattered clays | 1.0 |

## JOINT SET NUMBER (Jn)

| Number of joint sets | Joint set number (Jn) |
|---|---|
| Intact, no or few joint/fissures | 0.5 - 1.0 |
| One joint/fissure set | 2 |
| One joint/fissure set plus random | 3 |
| Two joint/fissure sets | 4 |
| Two joint/fissure sets plus random | 6 |
| Three joint/fissure sets | 9 |
| Three joint/fissure sets plus random | 12 |
| Four or more joint/fissure sets, random, heavily jointed, sugar cube, etc. | 15 |
| Multiple joint/fissure sets (crushed rock, earth-like) | 20 |

## STRESS REDUCTION FACTOR (SRF)

| | |
|---|---|
| Weakness zones intersecting pillar. | 10.0 |
| Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock (any depth). | 2.5 |
| Single weakness zones containing clay, or chemically disintegrated rock. | 2.5 |
| Multiple shear zones in competent rock (clay free), loose surrounding rock. | 7.5 |
| Single shear zones in competent rock (clay free). | 2.5 |
| Loose open joints, heavily jointed or 'sugar cube'. | 5.0 |

PROJECT NO. ........ DRAWN ........ REVIEWED ........ DATE ........

## DATA COLLECTION SHEET
## ROCK MASS CLASSIFICATION

Figure **A2**

| | |
|---|---|
| LOCATION: **Thompson Mine**<br>Level 1600 Face ______ Distance ______ | DATE<br>20 JAN. 89 |

| Rock Type | Strength | Sketch |
|---|---|---|
| **QUARTZITE** | | |

| Joint Data | Set 1 | Set 2 | Set 3 | Set 4 |
|---|---|---|---|---|
| Dip Direct/Dip Angle | 135/65 | 190/85 | NOT OBSERVED | — |
| Average Spacing (m) | 0.1 | 5 | — | — |
| Estimated No. of Joints/m$^3$ | | | | |

Q = 1.1

MRMR =

Avg. Vol. =

Shape

RQD ~30

### JOINT ALTERNATION NUMBER (Ja)

| Description of gouge | Joint alteration number (Ja) for joint separation (mm) | | |
|---|---|---|---|
| | <1.0[1] | 1.0-5.0[2] | >5.0[3] |
| Tightly healed, hard, nonsoftening impermeable filling | 1,2,3 | - | - |
| Unaltered joint walls, surface staining only | 1.0 | - | - |
| Slightly altered, non-softening, non-cohesive rock mineral or crushed rock filling | [✓ 2.0] | 4.0 | 6.0 |
| Non-softening, slightly clayey non-cohesive filling | 3.0 | [✓ 6.0] 2 | 10.0 |
| Softening or low friction clay mineral coatings and small quantities of swelling clays | 4.0 | 8.0[5] | 13.0 |
| Shattered or micro-shattered (swelling) clay gouge, with or without crushed rock | 5.0 | 10.0[4] | 18.0 |

### JOINT ROUGHNESS NUMBER (Jr)

| Joint Separation | Condition of Joint | Joint Roughness Number (Jr) |
|---|---|---|
| Joints/fissures tight or losing during excavation | Discontinuous joint/fissures | 3.0 |
| | Rough or irregular, indulating | 3.0 |
| | Smooth indulating | 2.0 |
| | Rough or irregular, planar | 1.2 |
| | Smooth planar | [✓ 1.0] 2. |
| | Slickensided planar | 0.5 |
| Joints/fissures open and remain open during excavation | Joints/fissures either open or containing relatively soft gouge or sufficient thickness to prevent joint/fissure wall contact upon excavation. | 1.0 |
| | Shattered or micro-shattered clays | 1.0 |

### JOINT SET NUMBER (Jn)

| Number of joint sets | Joint set number (Jn) |
|---|---|
| Intact, no or few joint/fissures | 0.5 - 1.0 |
| One joint/fissure set | 2 |
| One joint/fissure set plus random | 3 |
| Two joint/fissure sets | 4 |
| Two joint/fissure sets plus random | 6 |
| Three joint/fissure sets | [✓ 9] |
| Three joint/fissure sets plus random | 12 |
| Four or more joint/fissure sets, random, heavily jointed, sugar cube, etc. | 15 |
| Multiple joint/fissure sets (crushed rock, earth-like) | 20 |

### STRESS REDUCTION FACTOR (SRF)

| | |
|---|---|
| Weakness zones intersecting pillar. | 10.0 |
| Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock (any depth). | 2.5 |
| Single weakness zones containing clay, or chemically disintegrated rock. | 2.5 |
| Multiple shear zones in competent rock (clay free), loose surrounding rock. | 7.5 |
| Single shear zones in competent rock (clay free). | 2.5 |
| Loose open joints, heavily jointed or 'sugar cube'. | 5.0 |

PROJECT NO. .......... DRAWN .......... REVIEWED .......... DATE ..........

# APPENDIX B

Standard Procedure for Collection of Geotechnical Data
from Detail Line Surveys

| | |
|---|---|
| B1 to B8 | Detail Line Surveys & Figures |

# DETAIL LINE SURVEYS

## Method of Survey

The detail line method is a systematic sampling technique used to characterize rock joint properties (Call, 1972; Savely, 1972; and Piteau, 1973). Ideally, the sites are randomly selected and the lines have random orientations.

At the selected mapping site, a tape is stretched along the rock exposure to intersect the greatest number of joints. The tape is used as a reference line. The mapping zone is parallel to the tape and extends approximately 3 feet above and below the reference line. The fractures falling within the mapping zone are described using a standard format. A typical record sheet is shown in Figure 1. On this sheet the following information is recorded:

Joint Data (means any type of discontinuity)

| | | |
|---|---|---|
| Column 1. | NO. | The number of the discontinuity from the start of the detailed line survey. |
| 2. | DIST. | Distance along tape where the discontinuities or their projections intersect the tape. |
| 3. | TYPE | Type of discontinuity. At site includes: (See also Figure 2).<br>B = Bedding<br>JS = Joint Set<br>SJ = Single Joint<br>DI = Dike<br>CT = Contact<br>EX = Extremely weathered zone<br>EA = Extremely altered zone<br>FL = Foliation<br>S? = Undefined Surface<br>FT = Shear Plane (Fault)<br>SZ = Shear Zone (Fault)<br>CZ = Crushed Zone (Fault) |
| 4. | DIP DIR. | Direction of dip measured from Thompson Mine North |

| | | |
|---|---|---|
| 5. | DIP ANG. | Angle of dip measured from the horizontal. |
| 6. | SIZ. | Length in metres. |
| 7. | END | Can ends be seen?<br>2 = Both ends disappear from site or are covered.<br>1 = One end disappears from site.<br>0 = Both ends terminate within site. |
| 8. | RUFF | Roughness of joint surface as measured downdip with a carpenter's comb and compared to roughness profiles as defined by Barton (1977). (See Figure 3). |
| 9. | INFL. | Use Mines Exploration Standard abbreviations. |
| 10. | APPET. | Apparent width, as measured, or estimated.<br>Very tight = VT 0. - 0.1 mm<br>Tight = T 0.1 - 0.25 mm<br>Partly Open = PO 0.25 - 0.5 mm<br>Open = O 0.5 - 2.5 mm<br>Moderately Wide = MW 2.5 - 10 mm |
| 11. | FILL HRDN. | Fill hardness. The following scale is from the International Society of Rock Mechanics (1977). |

| Symbol | Description | Field Identification |
|---|---|---|
| S1 | Very soft soil or or rock. | Easily molded by fingers; shows distinct heel marks. |
| S2 | Soft soil or rock. | Molded with strong pressure from fingers; shows faint heel marks. |
| S3/R0 | Firm soil or extremely weak rock. | Very difficult to mold with fingers, indented with finger nail; difficult to cut with hand spade. |
| S4/R1 | Stiff soil or very weak rock. | Cannot be molded with fingers; crumbles under blow of hammer, or can be peeled with knife. |
| S5/R2 | Very stiff soil or weak rock. | Very tough, difficult to move with hand pick; peeled by knife with difficulty, indented by blow of hammer. |
| R3 | Medium-strong rock. | Cannot be scraped with knife; fractured with single blow of hammer. |
| R4 | Strong rock. | Requires more than single blow of hammer to fracture. |
| R5 | Very strong rock. | Requires many blows of hammer to fracture rock. |
| R6 | Extremely strong rock. | Can only be chipped by repeated blows of hammer. |

| | | |
|---|---|---|
| Column 12. | WTR. | Water condition along discontinuity.<br>1 D = dry<br>2 M = moist<br>3 W = wet, i.e., dripping<br>4 F = flow, continuous stream |
| 13. | WAV. | Waviness. Descriptive Classification<br>P = Planar<br>C = Curved<br>R = Rough or stepped<br>U = Undulating<br>(See Figure 4 for more complete explanation) |
| 14. | LITH. | Lithology of rock type at the point of discontinuity. Use Mines Exploration Standard abbreviations. |
| 15. | ROCK HRDN. | Hardness as based on the scale of R values given under Column 11. |
| 16. & 17. | SPARE | Depending upon the geologic environment, other factors related to discontinuities may be considered significant and may be coded. The following is however, recommended:- |
| 16. | MINIMUM DIP | The difference between minimum and maximum dip of the discontinuity within the mapping zone, measured in degrees. |
| 17. | | Numeric cross reference to notes made in field book. |

INDENTIFICATION:

LOCATION:

LOGGED BY: DATE: NO. OBSERVATIONS:

FORMATION:

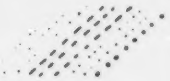
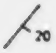
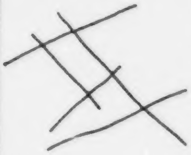
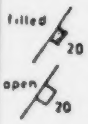
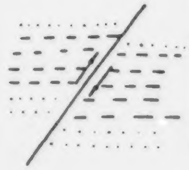
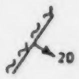
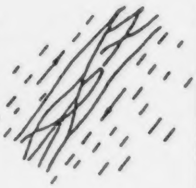
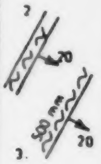
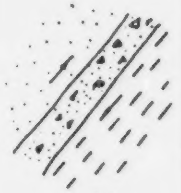
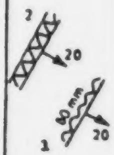
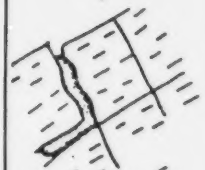
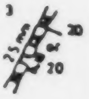
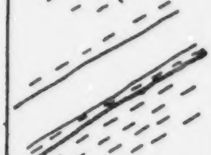
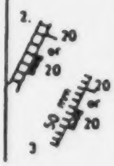
| JOINT MEASUREMENT SURVEY DATA: | NORTHING | EASTING | ELEVATION |
|---|---|---|---|
| START OF SURVEY LINE: | | | |
| END OF SURVEY LINE: | | | |

REMARKS:

| NO. | DIST. | TYPE | DIP DIR. | DIP ANG. | SIZE | END | RUFF | INFL 1 2 | APPET WIDTH | FILL HRON | WTR | WAV | LITHO | HDN | SPARE | SPARE |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|

RECORD SHEET

**FIGURE 1**

| APPEARANCE | NAME | SYMBOL | DESCRIPTION |
|---|---|---|---|
|  | Bedding |  | Layered or parallel arrangement of mineral grains, giving rise to planar anisotropy in the substance. |
|  | Joint |  | Discontinuity or crack across which rock usually has little tensile strength, may be planar, curved or irregular and open or filled by soil or rock substance which acts as cement; surfaces may be rough or smooth |
|  | Shear Plane (fault)[1] |  | Near planar surface along which movement has taken place; surfaces may be polished or slickensided and may have a film or coating of CLAY up to 2mm thick. |
|  | Sheared Zone / Seam (fault)[1] |  | Zone, with roughly parallel almost planar boundaries of rock substance cut by closely spaced (often < 50mm) shear planes and/or joints. The shear planes are usually smooth or slickensided and curved, intersecting to divide the mass into lenticular or wedge shaped blocks. Rock properties |
|  | Crushed Zone / Seam (fault)[1] |  | Zone with roughly parallel almost planar boundaries, composed of disoriented, usually angular, fragments of the host rock substance. The fragments may be of CLAY, SILT, SAND, or GRAVEL sizes or mixtures of these. Soil properties |
|  | Infill Zone / Seam |  | Zone of any shape but commonly with roughly parallel, planar boundaries. Formed by migration of soil into rock cavity or defect. |
|  | Extremely Weathered Zone / Seam |  | Zone of any shape but commonly with roughly parallel, planar boundaries often with gradational boundaries. Formed by weathering of the rock substance in place. |

NOTES

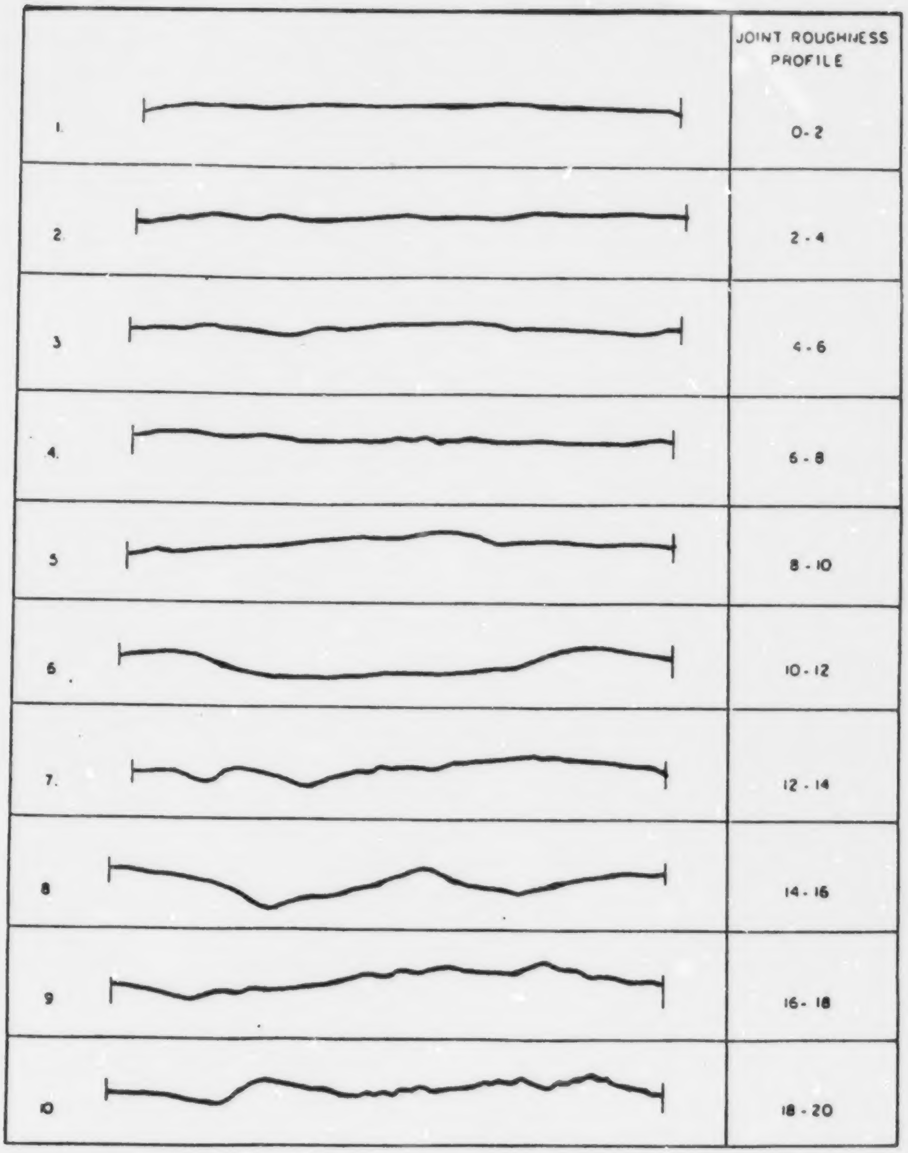
1. In the geological sense
2. To scale
3. Thickness indicated.

THOMPSON ROCK MECHANICS STUDY

FIGURE No. 2

## ROUGHNESS PROFILES

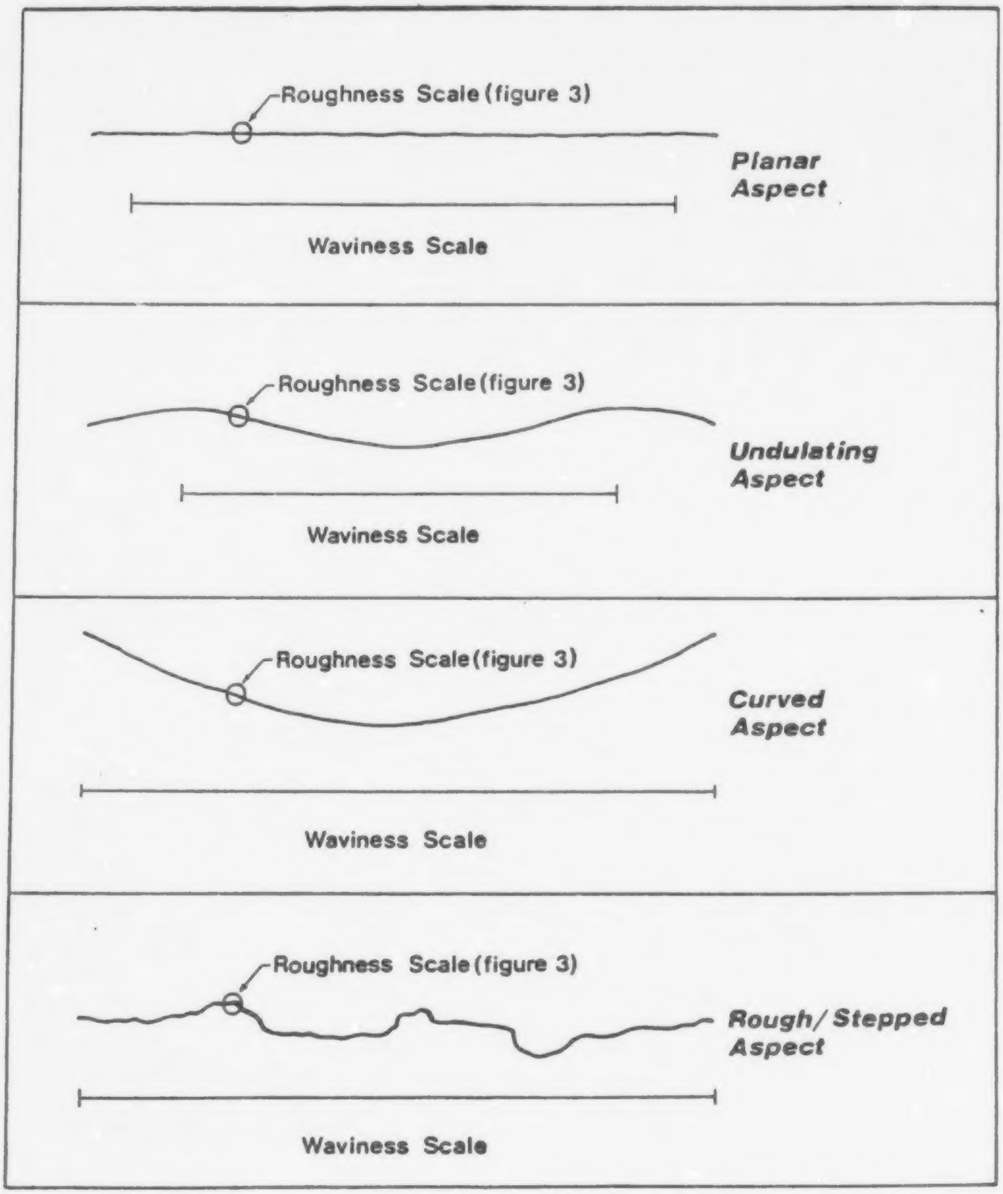
(AFTER BARTON, 1977)


JOINT ROUGHNESS PROFILE
1.
0-2
2
2-4
3
4-6
4.
6-8
5
8-10
6
10-12
7.
12-14
8
14-16
9
16-18
10
18-20

THOMPSON ROCK MECHANICS STUDY

FIGURE No. 3

## CLASSIFICATION OF JOINT WAVINESS


Roughness Scale (figure 3)
Planar Aspect
Waviness Scale
Roughness Scale (figure 3)
Undulating Aspect
Waviness Scale
Roughness Scale (figure 3)
Curved Aspect
Waviness Scale
Roughness Scale (figure 3)
Rough/ Stepped Aspect
Waviness Scale

THOMPSON ROCK MECHANICS STUDY

FIGURE No. 4

# APPENDIX C

## Standard Procedure for Collection of Geotechnical Data from Scan Line Surveys

| | |
|---|---|
| C1 to C4 | Suggested Method for Scanline Survey; Figures & Table |
| C5 to C14 | Program Scanline |
| C15 to C27 | Scanline Program Source Listing |

# SUGGESTED METHOD FOR SCANLINE SURVEY

Equipment Required: Clino Rule, Douglas protractor with orienting arm, 30m measuring tape, grease crayon, patience, recording sheets.

Method:

1. Select survey site. Ideally for every site 3 mutually perpendicular scanlines should be run; one along strike; one at right angles to strike; and one vertically. Due to access restrictions,however, this may not always be possible.

2. Stretch tape along selected survey area and record distance at the end of the tape. Mark ends for possible future reference. Note approximate location on level plan, and cross-referenced on recording sheet. Generally, it will not be necessary for survey to precisely locate the scanline, unless a specific geologic feature or structure is intersected (eg. a fault).

3. The tape marks the centre of a linear sampling window (see Figure C1) defined by a band 0.5m above and 0.5m below the tape. Only those discontinuities that fall within this window are to be measured.

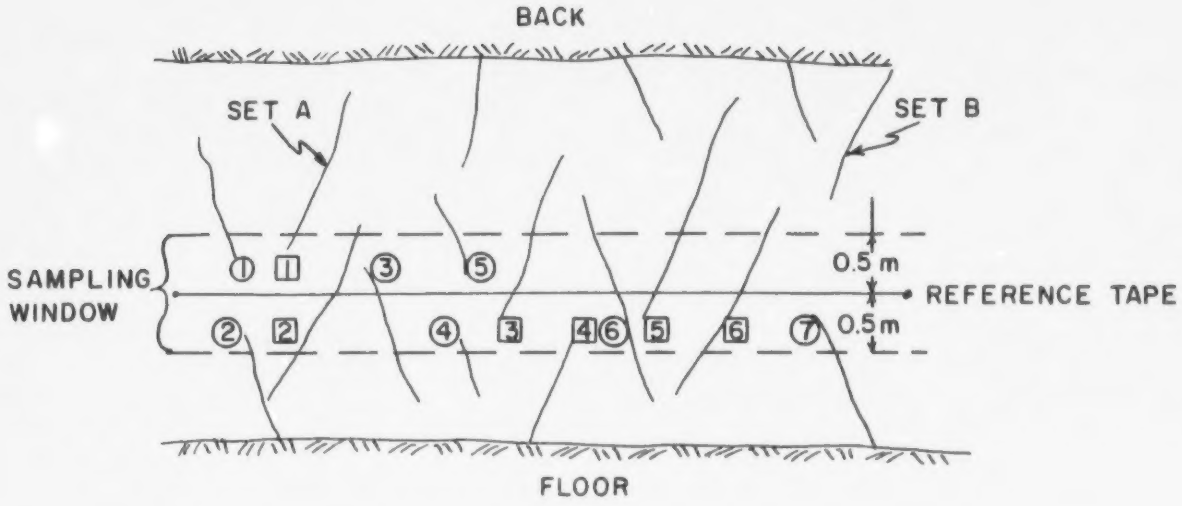
4. Inspect the sampling window to identify the number of joint sets and any random joints that exist within the sampling window; that is look first!

5. Start with the most frequently occurring joint set. Measure the dip and dip direction of the first joint of this set that interesects the sampling window. Move along the sampling window counting all joints with similar orientations until there is an evident change in the orientation. Record this number. Measure the dip and dip direction of the joint that occurs at the change and continue along the tape, counting those joints with similar orientations. Repeat this process until the end of the line is reached.

6. At measurement of dip and dip direction the surface condition of the joint surface should be recorded using either the carpenters comb to estimate a Joint Roughness Coefficient (JRC) with the aid of the profiles shown in Figure C2 or the classification outlined in Table C1. Whatever method selected should be clealy indicated.

7. Repeat steps 5) and 6) for the remaining joint sets identified and for any random joints.

FIGURE C-1 SAMPLING

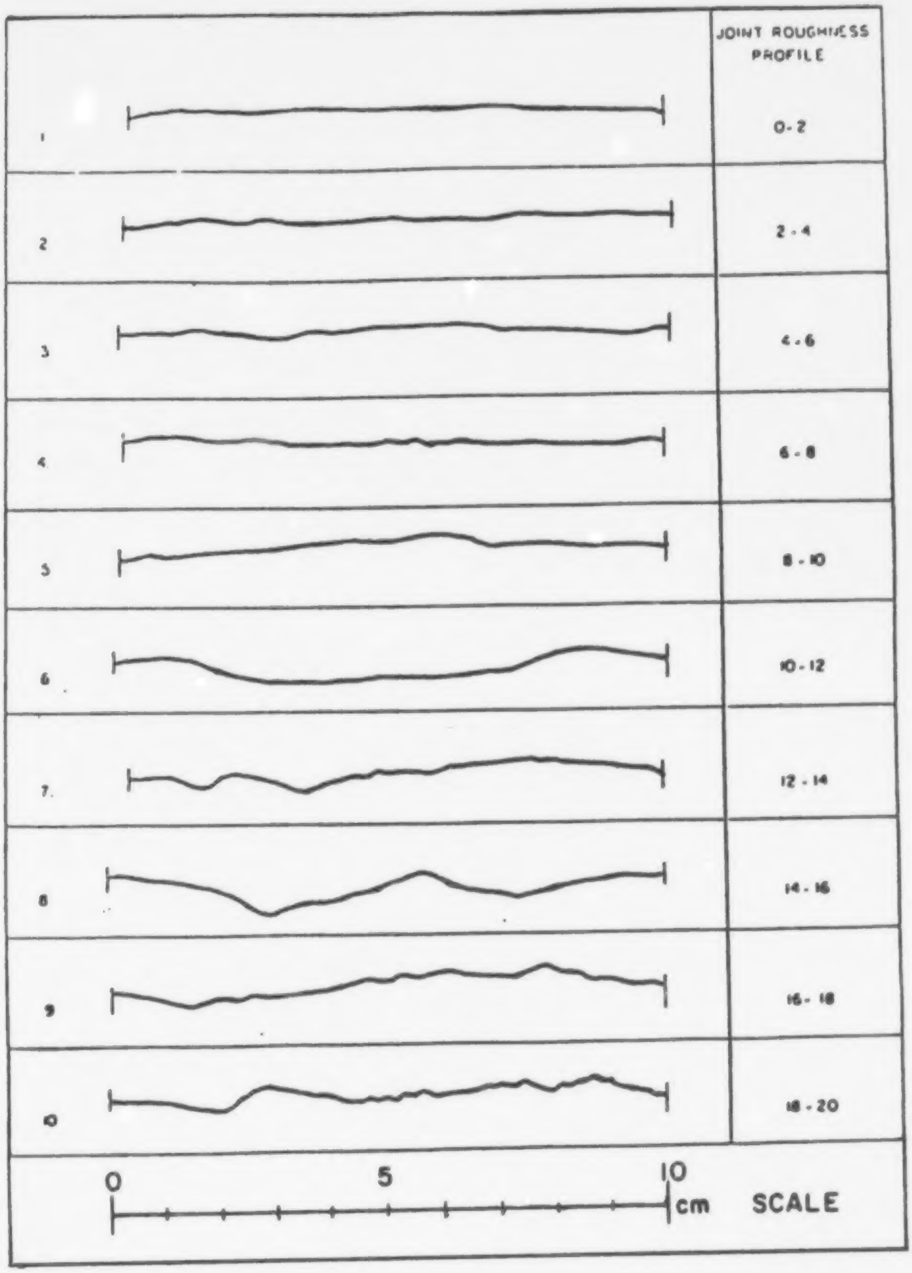

BACK
SET A
SET B
SAMPLING WINDOW
①
1
③
⑤
0.5 m
REFERENCE TAPE
②
2
④
3
4
⑥
5
6
⑦
0.5 m
FLOOR

ONLY COUNT THOSE JOINTS THAT FALL WITHIN SAMPLING WINDOW

For example window intersects 6 joints belonging to set A and 7 joints belonging to set 8.

## ROUGHNESS PROFILES

Figure C.2


JOINT ROUGHNESS PROFILE
1
0-2
2
2-4
3
4-6
4
6-8
5
8-10
6
10-12
7.
12-14
8
14-16
9
16-18
10
18-20
0
5
10
cm
SCALE

PROJECT NO. ............ DRAWN ............ REVIEWED ............ DATE ............

## Table C1

## Joint Roughness

| Condition of Joint | Joint Roughness Number |
|---|---|
| Discontinuous | 4.0 |
| Rough or Irregular Undulating | 3.0 |
| Smooth Undulating | 2.0 |
| Slickensided Undulating | 1.5 |
| Rough or Irregular Planar | 1.5 |
| Smooth Planar | 1.0 |
| Slickensided Planar | 0.5 |

# PROGRAM SCANLINE

## PROGRAM SCANLINE

Purpose: To estimate the true (i.e. perpendicular to mean orientation) joint spacing from the results of scanline surveys (see Appendix C for method of conducting scanline surveys). The program takes into account mean orientation of the defined joint sets, the number of joints belonging to each of the defined sets encountered by the scanline, corrects for directional bias and coalesces the results from different scanlines to give an estimated true mean spacing for each of the defined joint sets.

Input: Input consists of administrative data, the number of scanlines to be analysed their orientation, the number of joint sets, and the number of joints encountered within each joint set along the different scanlines.

Output: Output consists of a table showing the calculated joint frequencies and spacings for each joint set encountered within each scanline. A summary of all the results is also presented.

Running: The program is entered through the underground series of programs:

1) First type UG. A disclaimer will appear on the screen, Figure 1. A menu will then follow, Figure 2. Select item A, Rock Quality Analysis.

2) The Rock Quality Menu will appear on the screen, Figure 3. Select item B, Scanline Program. A header sheet will then follow.

3) A series of administrative questions will be asked regarding site, location and rock type as shown in Figure 4. The number of scanlines and number of joint sets for which data will be entered is then requested, Figure 5.

4) A table will then appear on the screen. Data on the dip and dip direction for each joint set can now be entered, Figure 6. Dip angles are entered in degrees measured down from the horizontal. Dip directions are entered in degrees measured clockwise from north.

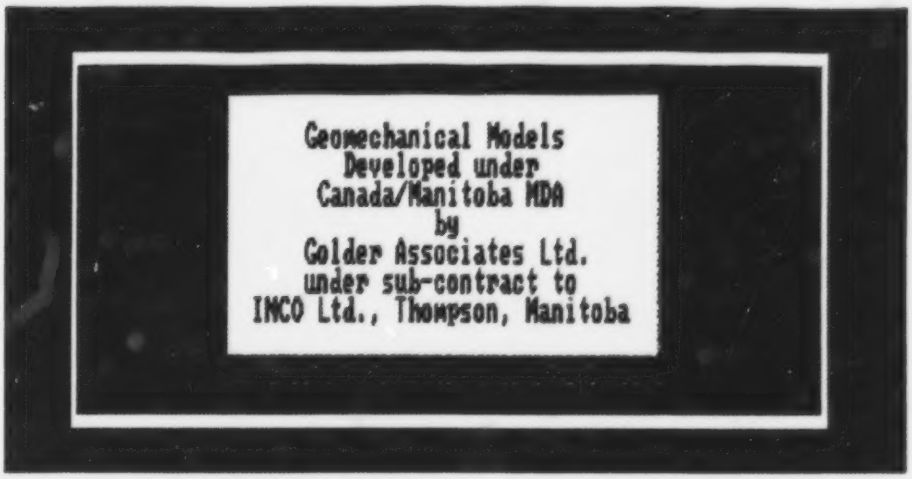
5) Once all the joint orientation information has been entered a second table will appear requesting scanline plunge, trend and length, Figure 7. Plunge and trend are both measured in degrees, down dip from horizontal for plunge and clockwise from north for trend. Once the appropriate information is typed in an overlay will appear asking for the number of intersection joints from joint set 1 made with the scanline, Figure 7. This process will be repeated until all the jointe set and scanline data is entered.

6) On completion of data entry a result table will appear, Figure 8 showing the frequency and mean spacing for the different joint sets

within each scanline, together with a overall mean and frequency for all the scanlines combined.

7) A hardcopy of the result can be obtained and further scanline surveys can be run, Figure 9.

2/am21605



Geomechanical Models
Developed under
Canada/Manitoba MDA
by
Golder Associates Ltd.
under sub-contract to
INCO Ltd., Thompson, Manitoba

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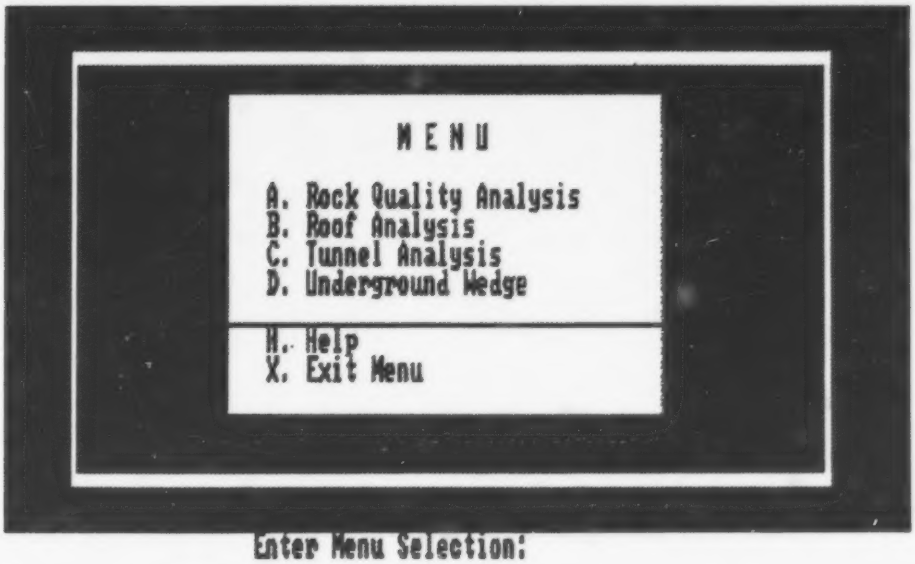
DISCLAIMER MESSAGE Figure 1

Press any key to continue . . .



M E N U

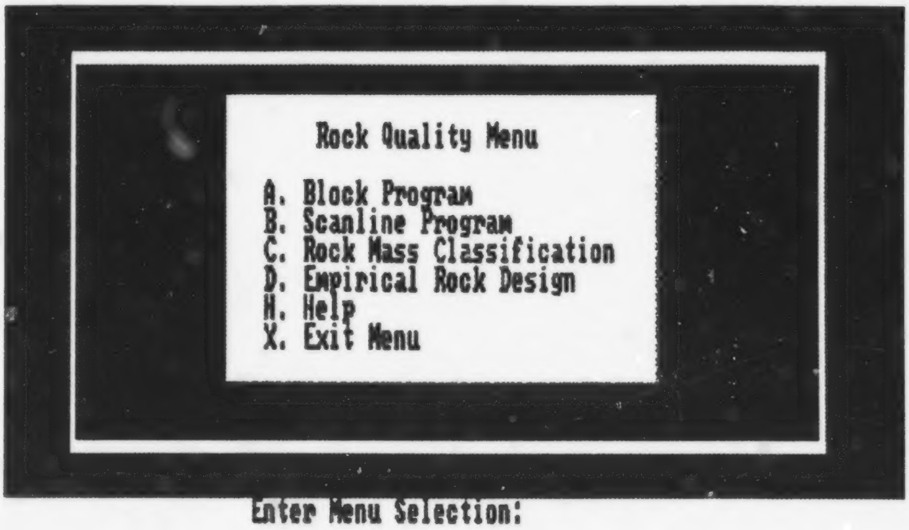
A. Rock Quality Analysis
B. Roof Analysis
C. Tunnel Analysis
D. Underground Wedge

H. Help
X. Exit Menu

Enter Menu Selection:

(Note programs A - Rock Quality, B - Roof Analysis, and C - Tunnel Analysis, have been developed as part of Task 1, VBM Design Guidelinese. Details of these programs given in Appendix entitled "Support Design" of the VBM guidelines).

MAIN MENU Figure 2



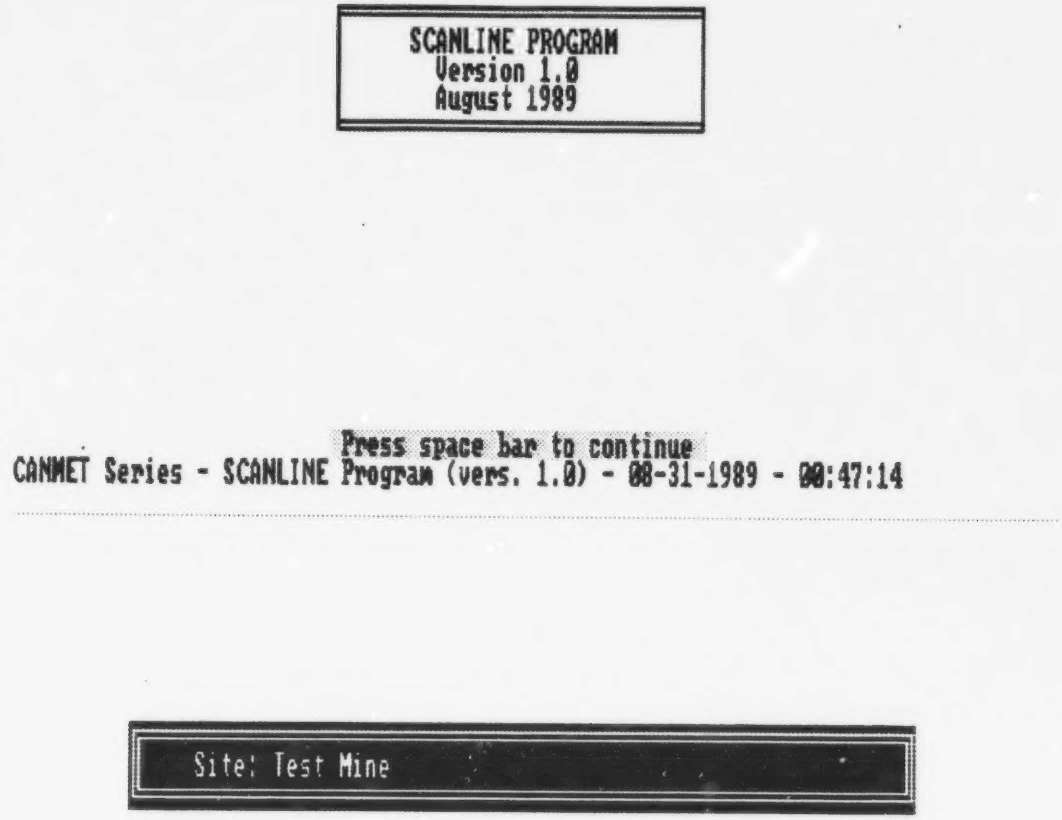
Rock Quality Menu

A. Block Program
B. Scanline Program
C. Rock Mass Classification
D. Empirical Rock Design
H. Help
X. Exit Menu

Enter Menu Selection:

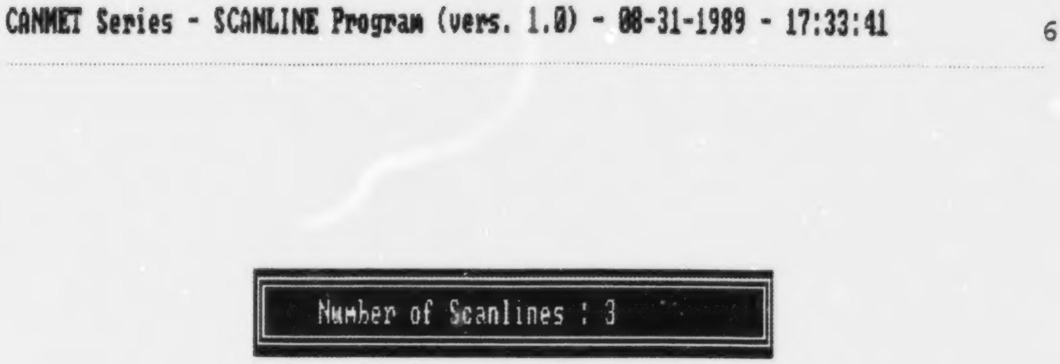
(Note program C - Rock Mass Classification and D - Empirical Rock Design have been developed as part of Task 1, VBM Design Guidelines. Details of the program are given in the Appendix entitled "Support Design" of the VBM guidelines.)

**ROCK QUALITY MENU** Figure 3

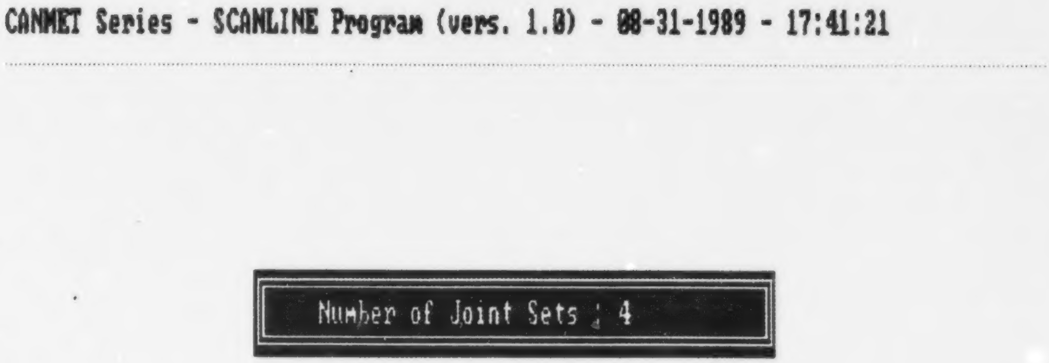


```
SCANLINE PROGRAM
  Version 1.0
  August 1989

                    Press space bar to continue
CANMET Series - SCANLINE Program (vers. 1.0) - 08-31-1989 - 00:47:14

     Site: Test Mine
```

**SCANLINE HEADER SHEET AND ADMINISTRATIVE DATA** Figure 4



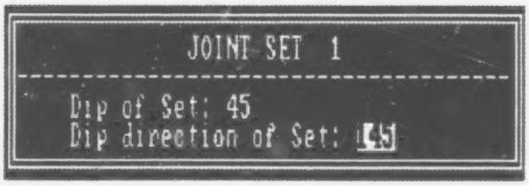
```
CANMET Series - SCANLINE Program (vers. 1.0) - 08-31-1989 - 17:33:41

Number of Scanlines : 3
```



```
CANMET Series - SCANLINE Program (vers. 1.0) - 08-31-1989 - 17:41:21

Number of Joint Sets : 4
```

**SCANLINE AND JOINT DATA** Figure 5

CANMET Series - SCANLINE Program (vers. 1.0) - 08-31-1989 - 17:34:57

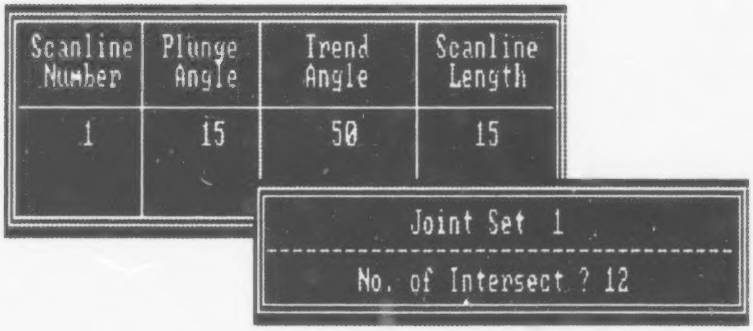


JOINT SET 1

Dip of Set: 45
Dip direction of Set: 45

JOINT SET ORIENTATIONS Figure 6

CANMET Series - SCANLINE Program (vers. 1.0) - 08-31-1989 - 17:36:45

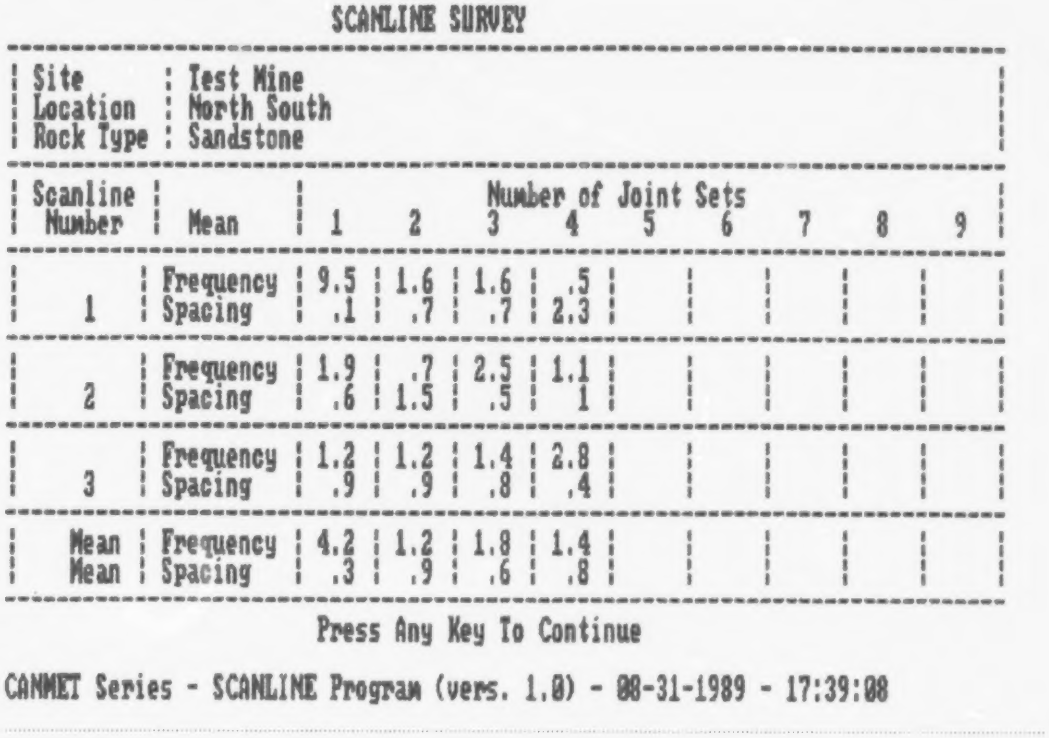


| Scanline Number | Plunge Angle | Trend Angle | Scanline Length |
|---|---|---|---|
| 1 | 15 | 50 | 15 |

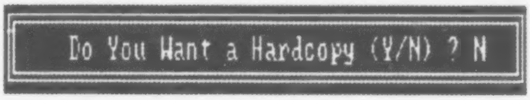
Joint Set 1

No. of Intersect ? 12

SCANLINE ORIENTATION AND NUMBER JOINTS INTERSECTION Figure 7

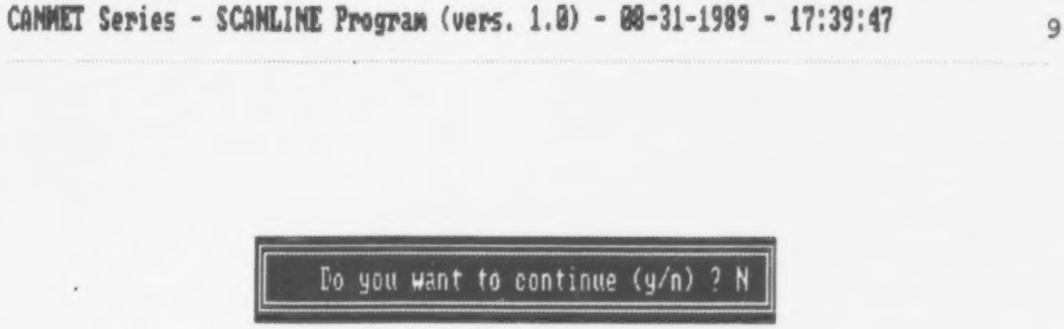


```
                          SCANLINE SURVEY
-------------------------------------------------------------------------
¦ Site      : Test Mine                                                 ¦
¦ Location  : North South                                               ¦
¦ Rock Type : Sandstone                                                 ¦
-------------------------------------------------------------------------
¦ Scanline ¦           ¦                Number of Joint Sets            ¦
¦  Number  ¦   Mean    ¦  1     2     3     4     5     6     7     8     9 ¦
-------------------------------------------------------------------------
¦          ¦ Frequency ¦ 9.5 ¦ 1.6 ¦ 1.6 ¦  .5 ¦     ¦     ¦     ¦     ¦     ¦
¦    1     ¦ Spacing   ¦  .1 ¦  .7 ¦  .7 ¦ 2.3 ¦     ¦     ¦     ¦     ¦     ¦
-------------------------------------------------------------------------
¦          ¦ Frequency ¦ 1.9 ¦  .7 ¦ 2.5 ¦ 1.1 ¦     ¦     ¦     ¦     ¦     ¦
¦    2     ¦ Spacing   ¦  .6 ¦ 1.5 ¦  .5 ¦   1 ¦     ¦     ¦     ¦     ¦     ¦
-------------------------------------------------------------------------
¦          ¦ Frequency ¦ 1.2 ¦ 1.2 ¦ 1.4 ¦ 2.8 ¦     ¦     ¦     ¦     ¦     ¦
¦    3     ¦ Spacing   ¦  .9 ¦  .9 ¦  .8 ¦  .4 ¦     ¦     ¦     ¦     ¦     ¦
-------------------------------------------------------------------------
¦     Mean ¦ Frequency ¦ 4.2 ¦ 1.2 ¦ 1.8 ¦ 1.4 ¦     ¦     ¦     ¦     ¦     ¦
¦     Mean ¦ Spacing   ¦  .3 ¦  .9 ¦  .6 ¦  .8 ¦     ¦     ¦     ¦     ¦     ¦
-------------------------------------------------------------------------
                      Press Any Key To Continue

CANMET Series - SCANLINE Program (vers. 1.0) - 08-31-1989 - 17:39:08
```

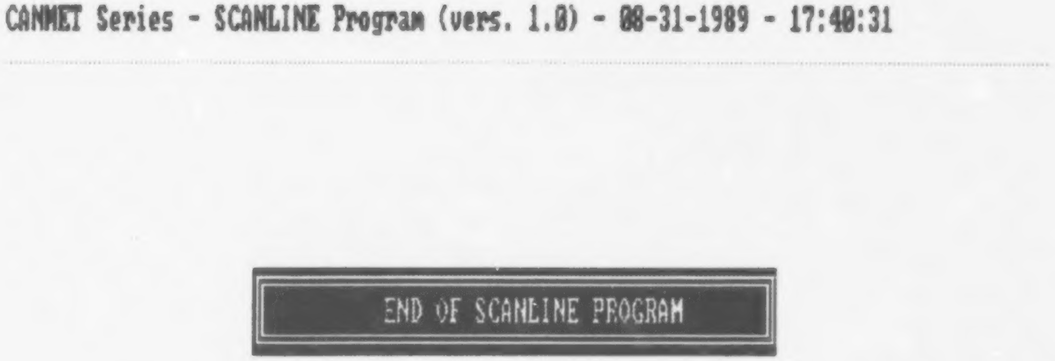


```
Do You Want a Hardcopy (Y/N) ? N
```

**TABLE OF RESULTS** Figure 8



```
CANMET Series - SCANLINE Program (vers. 1.0) - 08-31-1989 - 17:39:47

                Do you want to continue (y/n) ? N
```



```
CANMET Series - SCANLINE Program (vers. 1.0) - 08-31-1989 - 17:40:31

                    END OF SCANLINE PROGRAM
```

**PROGRAM CONCLUSION** Figure 9

# SCANLINE PROGRAM SOURCE LISTING

```
 DECLARE SUB GET.IMAGE.1 (X.START!, Y.START!, X.END!, Y.END!)
 DECLARE SUB PUT.IMAGE.1 (X.START!, Y.START!, X.END!, Y.END!)
 DECLARE SUB PAUSE ()
 DECLARE SUB GetConfig ()
 DECLARE SUB SetConfig (mode AS INTEGER)
 DECLARE SUB GBOXED (A!, B!, W!, D!)
 DECLARE SUB GNUMB (X!, Y!, X$, NOCHR%, B$, FG, BG)
 DECLARE SUB BOXED (X!, Y!, W!, D!, FG!, BG!)
 1 DEFINT I-N: DEFDBL S-T, V
 REM $DYNAMIC
  COMMON SHARED ISCRN.TYPE, ISCRN.COLOR, xscr1, yscr1, xscr2, yscr2
  COMMON SHARED BITS.PER.CHAR, BITS.PER.LINE, X1, Y1, X2, Y2, X3, Y3
  COMMON SHARED bin(), NPLANES, BITS.PER.PLANE, NSCAN
  COMMON SHARED VOL(), vmin, vmax
  COMMON SHARED IMAGE.1$()
  COMMON SHARED XSCRN.COL()

  DIM SMEAN(3), SPACING$(3), DIP$(3), DIPDIR$(3), HOR(4)
  DIM SPACING(3), DIP(3), DIPDIR(3), SSL(1), SSI(1), SSS(1)
  DIM MXCOL(10), inpos(10, 10), INFLD(10, 10), INTYP$(10, 10)
  DIM DA$(4, 100), SMEAN.FREQ(10), SMEAN.SPACING(10), SUM.SLAMDA(10)
  DIM SFREQ(10, 100), SSPACING(10, 100)
  DIM NSECT$(10, 100), ALPHAJ$(10), BETAJ$(10), ALPHAJ(10), BETAJ(10)

' General purpose constants
 CONST PI = 3.14159
 CONST TRUE = -1, FALSE = 0
 CONST BACK = 0
 CONST TROW = 24, TCOL = 60

' Constants for best available screen mode
 CONST VGA = 12
 CONST MCGA = 13
 CONST EGA256 = 9
 CONST EGA64 = 8
 CONST MONO = 10
 CONST HERC = 3
 CONST CGA = 2


' User-defined type to hold information about the mode
TYPE Config
  Scrn    AS INTEGER
  Colors  AS INTEGER
  Atribs  AS INTEGER
  XPix    AS INTEGER
  YPix    AS INTEGER
  TCOL     AS INTEGER
  TROW      AS INTEGER
END TYPE

DIM VC AS Config

' A palette of colors to paint with
DIM Pal(0 TO 300) AS LONG

' Error variables to check screen type
DIM InitRows AS INTEGER, BestMode AS INTEGER, Available AS STRING

 DATA 1
 DATA 4
 DATA 19,29,38,49
 DATA 2,2,3,3
 DATA "I","I","I","I"
 GOSUB 400
 GOTO 10
'----------------------------------------------------------------------
'                PRINT PAGE HEADER
'----------------------------------------------------------------------
2 SCREEN 0, 1: WIDTH 80: COLOR 7, 1: CLS
 PRINT "CANMET Series - SCANLINE Program (vers. 1.0) - " + DATE$ + " - " + TIME$
 COLOR 4, 1: PRINT STRING$(80, "_")
3 COLOR 7, 1: RETURN
'----------------------------------------------------------------------
'                PRINT PROGRAM HEADER
'----------------------------------------------------------------------
10 REM CANMET SCANLINE (Version 1.00) - PROGRAM HEADER
40 KEY OFF: SCREEN 0, 1: COLOR 5, 0, 0: WIDTH 80: CLS :
60 COLOR 7, 1: LOCATE 10, 25, 0: PRINT CHR$(213) + STRING$(27, 205) + CHR$(184)
75 LOCATE 11, 25, 0: PRINT CHR$(179) + "     SCANLINE PROGRAM      " + CHR$(179)
80 LOCATE 12, 25, 0: PRINT CHR$(179) + "       Version 1.0         " + CHR$(179)
90 LOCATE 13, 25, 0: PRINT CHR$(179) + "       August 1989         " + CHR$(179)
100 LOCATE 14, 25, 0: PRINT CHR$(212) + STRING$(27, 205) + CHR$(190)
120 COLOR 7, 4, 0: LOCATE 25, 25, 0: PRINT " Press space bar to continue ";
130 REM
140 CMD$ = INKEY$
150 IF CMD$ = "" THEN GOTO 140
170 IF CMD$ = " " THEN GOTO 190
180 GOTO 130
'----------------------------------------------------------------------
'                DATA ARRAY DEFINITION
'----------------------------------------------------------------------
190
' Get best configuration and set initial graphics mode to it
 GetConfig
 VC.Scrn = BestMode
 SetConfig VC.Scrn
 BITS.PER.CHAR = INT((VC.XPix + 1) / VC.TCOL)
 BITS.PER.LINE = INT((VC.YPix + 1) / VC.TROW)
 PLANES = VC.Atribs
 BITS.PER.PLANE = 1
 Scrn = VC.Scrn
 CMODE = VC.Colors
 XVU1 = 2 * BITS.PER.CHAR
 ZVU1 = 2 * BITS.PER.LINE
 XVU2 = (VC.TCOL - 2) * BITS.PER.CHAR
 ZVU2 = (VC.TROW - 2) * BITS.PER.LINE
 IEXIT = 0
 GOSUB 2
195 ISCRN.TYPE = VC.Scrn
  ISCRN.COLOR = 0
  xscr1 = 0
  yscr1 = 0
  xscr2 = VC.XPix
  yscr2 = VC.YPix
  DIM IMAGE.1$(VC.TCOL, VC.TROW)
  DIM XSCRN.COL(VC.TCOL, VC.TROW)
'----------------------------------------------------------------------
'                MAIN MENU
'----------------------------------------------------------------------
```

```
200 NSCAN = 0: NJTS = 0
   GOSUB 500: GOSUB 575: GOSUB 595
280 GOSUB 2: CALL BOXED(9, 20, 40, 3, 0, 7)
   LOCATE 10, 25: INPUT "Do you want to continue (y/n) "; an$
   an$ = LEFT$(an$, 1): IF an$ = "y" OR an$ = "Y" THEN GOTO 200
FINISH: REM ================< PROGRAM TERMINATION >=========================
 CLOSE : GOSUB 2
 IF VC.Colors > 0 THEN COLOR 9
 A = 10: B = 20: W = 40: D = 3: IOPT = 1
 CALL BOXED(A, B, W, D, FG, BG): IF VC.Colors > 0 THEN COLOR 4
 LOCATE 11, 30: PRINT "END OF SCANLINE PROGRAM"
 CALL PAUSE
 IF Scrn = 9 THEN COLOR 7
 END


' Error trap to make torus screen independent
VideoErr:
  SELECT CASE BestMode    ' Fall through until something works
    CASE VGA
        BestMode = MCGA
        Available = "12BD"
    CASE MCGA
        BestMode = EGA256
        Available = "12789"
    CASE EGA256
        BestMode = CGA
        Available = "12"
    CASE CGA
        BestMode = MONO
        Available = "A"
    CASE MONO
        BestMode = HERC
        Available = "3"
    CASE ELSE
        PRINT "Sorry. Graphics not available. Can't run Torus."
        END
  END SELECT
  RESUME

' Trap to detect 64K EGA
EGAErr:
  BestMode = EGA64
  Available = "12789"
  RESUME NEXT

' Trap to determine initial number of rows so they can be restored
RowErr:
  IF InitRows = 50 THEN
    InitRows = 43
    RESUME
  ELSE
    InitRows = 25
    RESUME NEXT
  END IF

  END
'------------------------------------------------------------
'                 LAYOUT INPUT FORM
'------------------------------------------------------------
```

```
400 :
   READ MXLEV
   FOR J = 1 TO MXLEV
   READ MXCOL(J)
   FOR I = 1 TO MXCOL(J): READ inpos(I, J): NEXT I
   FOR I = 1 TO MXCOL(J): READ INFLD(I, J): NEXT I
   FOR I = 1 TO MXCOL(J): READ INTYP$(I, J): NEXT I
   NEXT J
   RETURN

'-----------------------------------------------------------------------
'                    SPACING ROUTINES
'-----------------------------------------------------------------------
500 :
   GOSUB 2: A = 10: B = 10: W = 60: D = 3: FG = 0: BG = 7
   CALL BOXED(A, B, W, D, FG, BG)
   LOCATE 11, 15: LINE INPUT "Site: ", site$
   CALL BOXED(A, B, W, D, FG, BG)
   LOCATE 11, 15: LINE INPUT "Location : ", local$
   CALL BOXED(A, B, W, D, FG, BG)
   LOCATE 11, 15: LINE INPUT "Rock Type : ", rock.type$
   A = 10: B = 20: W = 40
   GOSUB 2: CALL BOXED(A, B, W, D, FG, BG)
   LOCATE 11, 25: INPUT "Number of Scanlines : ", NSCAN
   GOSUB 2: CALL BOXED(A, B, W, D, FG, BG)
   LOCATE 11, 25: INPUT "Number of Joint Sets : ", NJTS
   A = 11: B = 20: W = 40: D = 6: FG = 0: BG = 7
   RAD = 180! / 3.14159
   I = 0
I.LOOP:
   IF I < 0 THEN I = 0
   I = I + 1: IF I > NJTS THEN GOTO I.END
510 :
   GOSUB 2: CALL BOXED(A, B, W, D, FG, BG)
   LOCATE 12, 33: PRINT " JOINT SET "; I;
   LOCATE 13, 21: PRINT STRING$(38, "-");
   LOCATE 14, 25: PRINT "Dip of Set: ";
   X = 37: Y = 14: X$ = BETAJ$(I): NOCHR = 2: B$ = ""
   CALL GNUMB(X, Y, X$, NOCHR, B$, FG, BG)
   IF ASC(X$) = 27 OR ASC(B$) = 71 THEN I = I - 1: GOTO I.LOOP
   BETAJ$(I) = X$
   LOCATE 15, 25: PRINT "Dip direction of Set: ";
   X = 47: Y = 15: X$ = ALPHAJ$(I): NOCHR = 3: B$ = ""
   CALL GNUMB(X, Y, X$, NOCHR, B$, FG, BG)
   IF ASC(X$) = 27 OR ASC(B$) = 72 THEN GOTO 510
   ALPHAJ$(I) = X$
   ALPHAJ(I) = (VAL(ALPHAJ$(I)) + 180) / RAD
   BETAJ(I) = (VAL(BETAJ$(I)) = 90) / RAD
   GOTO I.LOOP
I.END:
   A = 4: B = 14: W = 43: D = NSCAN + 5
   GOSUB 2: CALL BOXED(A, B, W, D, FG, BG)
   LOCATE 5, 16: PRINT "Scanline  Plunge    Trend    Scanline ";
   LOCATE 6, 16: PRINT " Number   Angle    Angle     Length  ";
   LOCATE 7, 15
   IF ISCRN.TYPE <> 1 THEN
        PRINT STRING$(41, 196);
   ELSE
        PRINT STRING$(41, "-")
   END IF
   C1$ = CHR$(209): C2$ = CHR$(197): C3$ = CHR$(207)
   FOR I = 1 TO 3
   IF I = 1 THEN NCOL = 24
   IF I = 2 THEN NCOL = 33
   IF I = 3 THEN NCOL = 45
   FOR J = 4 TO NSCAN + 8
   LOCATE J, NCOL
   IF ISCRN.TYPE <> 1 THEN
        X$ = CHR$(179)
        IF J = 4 THEN X$ = C1$
        IF J = 7 THEN X$ = C2$
        IF J = NSCAN + 8 THEN X$ = C3$
   ELSE
        X$ = "|"
        IF J = 4 THEN X$ = "-"
        IF J = 7 THEN X$ = "-"
        IF J = NSCAN + 8 THEN X$ = "-"
   END IF
   PRINT X$;
   NEXT J: NEXT I
   NJTOT = 0
   NN = 0
   ISTART = 0
NN.LOOP:
   NN = NN + 1
   IF NN > NSCAN THEN GOTO NN.END
   ilev = 1
   X = inpos(1, ilev)
   Y = NN MOD 10: IF Y = 0 THEN Y = 10
   Y = Y + 7: LOCATE Y, X: PRINT RIGHT$("  " + STR$(NN), 2);
   IF ISTART = 0 THEN II = 1
II.LOOP:
   ISTART = 1
   II = II + 1
   IF II > 4 THEN GOTO II.END
   X = inpos(II, ilev)
   X$ = DA$(II, NN): B$ = "": NOCHR = INFLD(II, ilev)
   CALL GNUMB(X, Y, X$, NOCHR, B$, FG, BG)
   IF ASC(X$) = 27 OR ASC(B$) = 72 THEN
        II = II - 1
        NN = NN - 2
        IF NN < 0 THEN SOUND 440, 20: NN = 0: GOTO NN.LOOP
        GOTO NN.LOOP
   END IF
   IF ASC(B$) = 75 THEN
        II = II - 2
        IF II <= 0 THEN NN = NN - 2: II = 3: GOTO NN.LOOP
        GOTO II.LOOP
   END IF
   IF ASC(B$) = 77 THEN
        IF II + 1 > 4 THEN II = 3: GOTO NN.LOOP
        GOTO II.LOOP
   END IF
   IF ASC(B$) = 80 THEN
        II = II - 1
        GOTO NN.LOOP
   END IF
   DA$(II, NN) = X$
   GOTO II.LOOP
II.END:
   GOSUB 550
   ISTART = 0
   GOTO NN.LOOP
NN.END:
   NN = NSCAN
   GOSUB 2
   RETURN
'------------------------------------------------------------------
'                   INPUT JOINT DATA
'------------------------------------------------------------------
550 :
   X1 = 32: Y1 = 10
   X2 = 70: Y2 = Y1 + 4
   CALL GET.IMAGE.1(X1, Y1, X2, Y2)
   FOR II = 1 TO NJTS
   NJTOT = NJTOT + 1
   FOR JJ = 10 TO 14: LOCATE JJ, 33: PRINT STRING$(38, " "); : NEXT JJ
   A = 10: B = 33: W = 38: D = 5: CALL BOXED(A, B, W, D, FG, BG)
   LOCATE 11, 45: PRINT "Joint Set "; USING "##"; II
   LOCATE 12, 34: PRINT STRING$(36, "-")
   LOCATE 13, 40: INPUT " No. of Intersect "; NSECT$(II, NN)
   NEXT II
   FOR JJ = 10 TO 14: LOCATE JJ, 33: PRINT STRING$(38, " "); : NEXT JJ
   CALL PUT.IMAGE.1(X1, Y1, X2, Y2)
   RETURN
'------------------------------------------------------------------
'          ASSESSMENT DISCONTINUITY SPACING AND FRACTURE
'              FREQUENCY FROM SCANLINE SURVEYS
'------------------------------------------------------------------
575 :
   NJTOT = 0
   FOR J = 1 TO 10: SUM.SLAMDA(J) = 0: NEXT J
   FOR NN = 1 TO NSCAN
   ALPHAI = VAL(DA$(3, NN)) / RAD
   BETAI = VAL(DA$(2, NN)) / RAD
   SCLEN = VAL(DA$(4, NN))
   COSBI = COS(BETAI)
   SINBI = SIN(BETAI)
   FOR JJ = 1 TO NJTS
   NJ = VAL(NSECT$(JJ, NN))
   COSBJ = COS(BETAJ(JJ))
   SINBJ = SIN(BETAJ(JJ))
   COST = COS(ALPHAI - ALPHAJ(JJ)) * COSBI * COSBJ + SINBI * SINBJ
   SLAMDA = NJ / (SCLEN * ABS(COST))
   SUM.SLAMDA(JJ) = SUM.SLAMDA(JJ) + SLAMDA
   SFREQ(JJ, NN) = SLAMDA
   SSPACING(JJ, NN) = 1 / SFREQ(JJ, NN)
   NEXT JJ
   NEXT NN
   FOR JJ = 1 TO NJTS
   NTOT = 0
   FOR NN = 1 TO NSCAN
   IF VAL(NSECT$(JJ, NN)) > 0 THEN NTOT = NTOT + 1
   NEXT NN
   SMEAN.FREQ(JJ) = SUM.SLAMDA(JJ) / NTOT
   SMEAN.SPACING(JJ) = 1! / SMEAN.FREQ(JJ)
   NEXT JJ
   RETURN
'------------------------------------------------------------------
'                   PRINT SPACING RESULTS
```

```
'--------------------------------------------------------------------------------
595 :
  OPEN "SPACING.PRN" FOR OUTPUT AS #1
  COLOR 7, 1
  VERT$ = CHR$(124): HOR$ = CHR$(45)
  PRINT #1, TAB(26); "SCANLINE SURVEY"
  PRINT #1, STRING$(77, HOR$)
  PRINT #1, VERT$ + " Site      : "; site$; TAB(77); VERT$
  PRINT #1, VERT$ + " Location  : "; local$; TAB(77); VERT$
  PRINT #1, VERT$ + " Rock Type : "; rock.type$; TAB(77); VERT$
  PRINT #1, STRING$(77, HOR$)
  PRINT #1, VERT$ + " Scanline " + VERT$; TAB(23); VERT$; TAB(38); "Number of Joint Sets"; TAB(77);
VERT$
  PRINT #1, VERT$ + "  Number  " + VERT$ + "  Mean    " + VERT$ + " 1    2    3    4    5    6    7    8    9  " +
VERT$
  PRINT #1, STRING$(77, HOR$)
  FOR NN = 1 TO NSCAN
  d1$ = VERT$ + "          " + VERT$ + " Frequency " + VERT$
  d2$ = VERT$ + "    " + RIGHT$("  " + STR$(NN), 2) + "    " + VERT$ + " Spacing   " + VERT$
  FOR JJ = 1 TO 9
  IF SFREQ(JJ, NN) = 0! THEN
          FREQ$ = "    "
  ELSE
          FREQ = INT(SFREQ(JJ, NN) * 10 + .9)
          FREQ$ = RIGHT$("    " + STR$(FREQ / 10), 4)
  END IF
  IF SSPACING(JJ, NN) = 0! THEN
          SPAC$ = "    "
  ELSE
          SPAC = INT(SSPACING(JJ, NN) * 10 + .9)
          SPAC$ = RIGHT$("    " + STR$(SPAC / 10), 4)
  END IF
  d1$ = d1$ + FREQ$ + " " + VERT$
  d2$ = d2$ + SPAC$ + " " + VERT$
  NEXT JJ
  PRINT #1, d1$
  PRINT #1, d2$
  PRINT #1, STRING$(77, HOR$)
  NEXT NN
  d1$ = VERT$ + "    Mean " + VERT$ + " Frequency " + VERT$
  d2$ = VERT$ + "    Mean " + VERT$ + " Spacing   " + VERT$
  FOR JJ = 1 TO 9
  IF SMEAN.FREQ(JJ) = 0! THEN
          FREQ$ = "    "
  ELSE
          FREQ = INT(SMEAN.FREQ(JJ) * 10 + .9)
          FREQ$ = RIGHT$("    " + STR$(FREQ / 10), 4)
  END IF
  IF SMEAN.SPACING(JJ) = 0! THEN
          SPAC$ = "    "
  ELSE
          SPAC = INT(SMEAN.SPACING(JJ) * 10 + .9)
          SPAC$ = RIGHT$("    " + STR$(SPAC / 10), 4)
  END IF
  d1$ = d1$ + FREQ$ + " " + VERT$
  d2$ = d2$ + SPAC$ + " " + VERT$
  NEXT JJ
  PRINT #1, d1$
  PRINT #1, d2$
  PRINT #1, STRING$(77, HOR$)
```

```
   CLOSE #1
   OPEN "spacing.prn" FOR INPUT AS #1
   CLS
cont.on:
   LOCATE 25, 25: PRINT "Press Any Key To Continue";
   FOR I = 1 TO 24: IF EOF(1) THEN GOTO PAUSE.NOW
   LINE INPUT #1, screen.line$: PRINT screen.line$; : NEXT I
PAUSE.NOW:
   X$ = INKEY$:
   IF X$ = "" THEN
           GOTO PAUSE.NOW
   ELSE
           IF EOF(1) THEN GOTO 599
           GOTO cont.on
   END IF
599 :
   GOSUB 2
   CALL BOXED(9, 20, 40, 3, 0, 7)
   LOCATE 10, 25: INPUT "Do You Want a Hardcopy (Y/N) "; an$
   an$ = LEFT$(an$, 1): IF an$ = "n" OR an$ = "N" THEN GOSUB 2: RETURN
   LPRINT TAB(26); "JOINT SET DATA"
   LPRINT
   FOR JJ = 1 TO NJTS
   LPRINT "Joint Set "; USING "##"; JJ;
   LPRINT TAB(20); "Dip of Set = "; USING "##"; VAL(ALPHAJ$(JJ));
   LPRINT "         Dip direction = "; USING "###"; VAL(BETAJ$(JJ))
   NEXT JJ
   LPRINT
   LPRINT
   LPRINT TAB(26); "SCANLINE SURVEY"
   LPRINT STRING$(77, "-")
   LPRINT "| Site      : "; site$; TAB(77); "|"
   LPRINT "| Location  : "; local$; TAB(77); "|"
   LPRINT "| Rock Type : "; rock.type$; TAB(77); "|"
   LPRINT STRING$(77, "-")
   LPRINT "| Scanline|"; TAB(23); "|"; TAB(38); "Number of Joint Sets"; TAB(77); "|"
   LPRINT "|  Number |   Mean    | 1    2    3    4    5    6    7    8    9 |"
   LPRINT STRING$(77, "-")
   FOR NN = 1 TO NSCAN
   d1$ = "|         | Frequency |"
   d2$ = "|    " + RIGHT$("  " + STR$(NN), 2) + "   | Spacing   |"
   FOR JJ = 1 TO 9
   IF SFREQ(JJ, NN) = 0! THEN
            FREQ$ = "    "
   ELSE
            FREQ = INT(SFREQ(JJ, NN) * 10 + .9)
            FREQ$ = RIGHT$("    " + STR$(FREQ / 10), 4)
   END IF

   IF SSPACING(JJ, NN) = 0! THEN
            SPAC$ = "    "
   ELSE
            SPAC = INT(SSPACING(JJ, NN) * 10 + .9)
            SPAC$ = RIGHT$("    " + STR$(SPAC / 10), 4)
   END IF
   d1$ = d1$ + FREQ$ + " |"
   d2$ = d2$ + SPAC$ + " |"
   NEXT JJ
   LPRINT d1$
   LPRINT d2$
   LPRINT STRING$(77, "-")
   NEXT NN
   d1$ = "|   Mean   | Frequency |"
   d2$ = "|   Mean   | Spacing   |"
   FOR JJ = 1 TO 9
   IF SMEAN.FREQ(JJ) = 0! THEN
           FREQ$ = "    "
   ELSE
           FREQ = INT(SMEAN.FREQ(JJ) * 10 + .9)
           FREQ$ = RIGHT$("    " + STR$(FREQ / 10), 4)
   END IF
   IF SMEAN.SPACING(JJ) = 0! THEN
           SPAC$ = "    "
   ELSE
           SPAC = INT(SMEAN.SPACING(JJ) * 10 + .9)
           SPAC$ = RIGHT$("    " + STR$(SPAC / 10), 4)
   END IF
   d1$ = d1$ + FREQ$ + " |"
   d2$ = d2$ + SPAC$ + " |"
   NEXT JJ
   LPRINT d1$
   LPRINT d2$
   LPRINT STRING$(77, "-")
   LPRINT TAB(35); "Golder Associates"
   GOSUB 2
   RETURN

'------------------------------------------------------------------------
'                    TERMINATE PROGRAM
'------------------------------------------------------------------------
9000 REM
9010 CLOSE : A = 5: B = 20: W = 40: D = 3: FG = 0: BG = 7: CALL BOXED(A, B, W, D, FG, BG)
   LOCATE 6, 30: PRINT "END OF DATA INPUT";
9020 END
DEFINT A-H, O-Z

REM $STATIC
DEFSNG A-H, O-R, U, W-Z
DEFDBL S-T, V
SUB BOXED (X, Y, W, D, FG, BG) STATIC
'------------------------------------------------------------------------
'                 DRAW A DOUBLE-LINED BOX
'------------------------------------------------------------------------
COLOR FG, BG: LOCATE X, Y
PRINT CHR$(201) + STRING$(W - 2, 205) + CHR$(187);
  FOR L = X + 1 TO X + D - 2: LOCATE L, Y
    PRINT CHR$(186) + STRING$(W - 2, 32) + CHR$(186);
  NEXT L
LOCATE X + D - 1, Y: PRINT CHR$(200) + STRING$(W - 2, 205) + CHR$(188);
END SUB

REM $DYNAMIC
SUB GBOXED (A, B, W, D) STATIC
IF ISCRN.TYPE <> 1 THEN
        X = A: Y = B: LOCATE X, Y
        PRINT CHR$(201) + STRING$(W - 2, 205) + CHR$(187);
        FOR L = X + 1 TO X + D - 2: LOCATE L, Y
        PRINT CHR$(186) + STRING$(W - 2, 32) + CHR$(186);
        NEXT L
        LOCATE X + D - 1, Y: PRINT CHR$(200) + STRING$(W - 2, 205) + CHR$(188);
```

```
ELSE
        X = A: Y = B: LOCATE X, Y
        PRINT STRING$(W, "-");
        FOR L = X + 1 TO X + D - 2: LOCATE L, Y
        PRINT "|" + STRING$(W - 2, " ") + "|";
        NEXT L
        LOCATE X + D - 1, Y: PRINT STRING$(W, "-");
END IF
END SUB

REM $STATIC
DEFSNG I-N, S-T, V
SUB GET.IMAGE.1 (X.START, Y.START, X.END, Y.END) STATIC
' SAVE SCREEN IMAGE IN ASCII
        X.SIZE = X.END - X.START + 1
        Y.SIZE = Y.END - Y.START + 1
        FOR Y = Y.START TO Y.END
            FOR X = X.START TO X.END
                IMAGE.1$(X, Y) = CHR$(SCREEN(Y, X))
                XSCRN.COL(X, Y) = SCREEN(Y, X, 1)
            NEXT X
        NEXT Y

END SUB

DEFINT I-N
DEFDBL S-T, V
' ============================ GetConfig ==============================
'  Get the starting number of lines and the video adapter.
' =====================================================================
'

SUB GetConfig STATIC
SHARED InitRows AS INTEGER, BestMode AS INTEGER, Available AS STRING

  ' Assume 50 line display and fall through error
  ' until we get the actual number
  InitRows = 50
  ON ERROR GOTO RowErr
  LOCATE InitRows, 1

  ' Assume best possible screen mode
  BestMode = VGA
  Available = "12789BCD"

  ON ERROR GOTO VideoErr
  ' Fall through error trap until a mode works

  SCREEN BestMode
  ' If EGA, then check pages to see whether more than 64K
  ON ERROR GOTO EGAErr
  IF BestMode = EGA256 THEN SCREEN 8, , 1

  ON ERROR GOTO 0

  ' Reset text mode
  SCREEN 0, , 0
  WIDTH 80, 25

END SUB
```

```
SUB GNUMB (X, Y, X$, NOCHR, B$, FG, BG) STATIC
REM ================< NUMBER KEYS - SINGLE KEY ENTRY >=====================
IF LEN(X$) > 1 THEN C$ = RIGHT$(X$, NOCHR) ELSE C$ = STRING$(NOCHR, 32)
LOCATE Y, X: COLOR BG, FG: PRINT C$: X$ = "": B$ = " ": IFRST = 0
GNUMB1: A$ = INKEY$: IF A$ = "" THEN GOTO GNUMB1
IF LEN(A$) > 1 THEN X$ = " ": B$ = RIGHT$(A$, 1): GOTO GNUMB9
IF ASC(A$) = 8 THEN GOTO GNUMB3                ' Backspace key
IF ASC(A$) = 13 THEN GOTO GNUMB9               ' Return key
IF ASC(A$) = 27 THEN GOTO GNUMB8               ' Escape key
IF ASC(A$) = 46 THEN GOTO GNUMB2               ' Decimal key
IF ASC(A$) = 45 AND IFRST = 0 THEN GOTO GNUMB2      ' Minus key
IF ASC(A$) = 43 AND IFRST = 0 THEN GOTO GNUMB2      ' Plus key
IF ASC(A$) > 47 AND ASC(A$) < 58 THEN GOTO GNUMB2   ' Number key
IF ASC(A$) = 47 THEN IFRST = -1: GOTO GNUMB2        ' Slash key
GOTO GNUMB1
GNUMB2: X$ = X$ + A$:
IF LEN(X$) > NOCHR THEN SOUND 440, 20: X$ = LEFT$(X$, NOCHR): IFRST = IFRST - 1
LOCATE Y, X: PRINT X$; : IFRST = IFRST + 1
GOTO GNUMB1
GNUMB3: NL = LEN(X$): IF NL > 1 THEN X$ = LEFT$(X$, NL - 1) ELSE X$ = ""
LOCATE Y, X: PRINT C$; : LOCATE Y, X: PRINT X$; : GOTO GNUMB1
GNUMB8: X$ = A$: GOTO GEXIT
GNUMB9: LOCATE Y, X: COLOR FG, BG: PRINT STRING$(NOCHR, 32): LOCATE Y, X
PRINT RIGHT$(STRING$(NOCHR, 32) + X$, NOCHR);
GEXIT: END SUB

SUB PAUSE STATIC

CONT.PAUSE:

    X$ = INKEY$: IF X$ = "" THEN GOTO CONT.PAUSE

END SUB

DEFSNG A-H, O-Z
SUB PUT.IMAGE.1 (X.START, Y.START, X.END, Y.END)
'REPLACE SCREEN IMAGE FROM ARRAY
      X.SIZE = X.END - X.START + 1
      Y.SIZE = Y.END - Y.START + 1
      FOR Y = Y.START TO Y.END
            FOR X = X.START TO X.END
                  LOCATE Y, X
                  IF X > 56 OR Y > NSCAN + 8 THEN
                    COLOR 0, 1
                  ELSE
                     COLOR 0, 7
                  END IF

                  PRINT IMAGE.1$(X, Y);
            NEXT X
            COLOR 0, 7
      NEXT Y
      COLOR 0, 7

END SUB

DEFINT A-H, O-Z
' ========================== SetConfig ===========================
' Sets the correct values for each field of the VC variable. They
' vary depending on Mode and on the current configuration.
```

```
'===========================================================================
'
SUB SetConfig (mode AS INTEGER) STATIC
SHARED VC AS Config, BestMode AS INTEGER

  SELECT CASE mode
    CASE 1  ' Four-color graphics for CGA, EGA, VGA, and MCGA
        IF BestMode = CGA OR BestMode = MCGA THEN
          VC.Colors = 0
        ELSE
          VC.Colors = 16
        END IF
        VC.Atribs = 4
        VC.XPix = 319
        VC.YPix = 199
        VC.TCOL = 40
        VC.TROW = 25
    CASE 2  ' Two-color medium-res graphics for CGA, EGA, VGA, and MCGA
        IF BestMode = CGA OR BestMode = MCGA THEN
          VC.Colors = 0
        ELSE
          VC.Colors = 16
        END IF
        VC.Atribs = 2
        VC.XPix = 639
        VC.YPix = 199
        VC.TCOL = 80
        VC.TROW = 25
    CASE 3  ' Two-color high-res graphics for Hercules
        VC.Colors = 0
        VC.Atribs = 2
        VC.XPix = 720
        VC.YPix = 348
        VC.TCOL = 80
        VC.TROW = 25
    CASE 7  ' 16-color medium-res graphics for EGA and VGA
        VC.Colors = 16
        VC.Atribs = 16
        VC.XPix = 319
        VC.YPix = 199
        VC.TCOL = 40
        VC.TROW = 25
    CASE 8  ' 16-color high-res graphics for EGA and VGA
        VC.Colors = 16
        VC.Atribs = 16
        VC.XPix = 639
        VC.YPix = 199
        VC.TCOL = 80
        VC.TROW = 25
    CASE 9  ' 16- or 4-color very high-res graphics for EGA and VGA
        VC.Colors = 64
        IF BestMode = EGA64 THEN VC.Atribs = 4 ELSE VC.Atribs = 16
        VC.XPix = 639
        VC.YPix = 349
        VC.TCOL = 80
        VC.TROW = 25
    CASE 10 ' Two-color high-res graphics for EGA or VGA monochrome
        VC.Colors = 0
        VC.Atribs = 2
        VC.XPix = 319
```

# APPENDIX D

## Standard Procedure for Collection of Geotechnical Data from Drill Core

D1 to D3 Logging of Goetechnical Drill Holes
D4 to D7 Tables 1-4

# LOGGING OF GEOTECHNICAL DRILL HOLES

Column 1. DEPTH Length along core axis from collar of hole at end of run. Example: A run from 15.7 feet to 24.3 feet would be recorded as 24.3.

Column 2. WEATH/ALT Weathering or alteration of rock mass. If weathered or altered, shown by W or A in first block. This is modified by a number in second block representing the relative degree of chemical decomposition as described in Tables 1 and 2. Example: Cyrstalline rock (granitic) near surface in which feldspars have been clouded by partial alteration to clay and biotite has been partially altered to chlorite L2.

If the rock is pervasively altered such that through metamorphic lateration (retrogression) or surface affects the rock no longer exhibits the characteristics which would be expected of the rock, the principal type of alteration will be indicated in block one by a letter symbol.

A) Argillic or argillaceous; clay.

Q) Silification; pervasive alteration to silica.

S) Sericitization; sericite-quartz.

P) Potassic; potash feldspar, biotite, muscovite.

L) Low grade metamorphic alteration; epidotization, chloritization.

C) Calcareous; carbonatization.

G) Silication; metamorphic alteration to silicate minerals, i.e., silicate scarn.

I) Iron - Hematite.

The degree of alteration will be shown in block two by a number as described in the Table 2.

Column 3. ROCK TYPE Blocks 1 to 4 are to designate the rock type. The common types are listed below. For others use the Mines Exploration standard abbreviations.

| | |
|---|---|
| AMPT | Amphibolite |
| ARK | Arkose |
| CHRT | Chert |
| DLMT | Dolomite |
| DNT | Dunite |

| | |
|---|---|
| GN | Gneiss - if known use generic type:<br>ORGN - Orthogneiss<br>PRGN - Paragneiss |
| GRGN | Granite Gneiss |
| GYWK | Greywacke |
| HBLT | Hornblendite |
| IF | Iron Formation |
| MRBL | Marble |
| PEG | Pegmatite |
| PRDT | Peridotite |
| PXT | Pyroxenite |
| QTE | Quartzite |
| SRPT | Serpentinite |
| SKN | Skarn |
| SCH | Schist |
| MASU | Massive Sulphide |
| SUMX | Sulphide Matrix |

Use M as a prefix for mineralized rock type and in the case of a 4 letter abbreviation, delete the last letter, i.e., MPRD.

| | | |
|---|---|---|
| Column 4. | R.Q.D. | R.Q.D. is Rock Quality Designation as defined by Deere (1967). This measure of the percentage of the core run that is in pieces of 4 inches or greater in length. |
| Column 5. | TOTAL CORE RECOVERY | The footage of core recovered in a run expressed as a percentage of the total run. |
| Column 6. | DIP AND CHAR. OF FRACTURES | Record the number of fractures that dip within range, i.e., 90° - 70°, in run. Dip is measured relative to core axis; 0° = parallel to core axis, 90° = at right angles. |
| Column 7. | | Roughness measure with carpenters comb. Compare with figures on roughness profile attached. |
| Column 8. | ALTERATION /FILLING | Define characteristics of fracture:<br>1. Tight<br>2. Open<br>3. Stained<br>4. Filled<br>If stained, type of stain:<br>1. Iron<br>2. Clay<br>If filled, type of filling: |

| | | |
|---|---|---|
| | | 1. Quartz<br>2. Carbonate<br>3. Clay - chlorite, biotite, anthophyllite, tremolite (weak)<br>4. Mineralized - gangue plus sulphide or other minerals. |
| Column 9. | TYPE OF GENETIC CLASSIFICATION | 1. Bedding<br>2. Foliation<br>3. Flow boundary or lamination<br>4. Contact<br>5. Cleavage<br>6. Shear Plane (Fault)<br>7. Shear Zone (Fault)<br>8. Crushed Zone (Fault)<br>9. Joint |
| Column 10. | STRENGTH | Classification of cohesive soil or rock hardness according to Table 3. |

# TABLE 1 - WEATHERING

| TERM | GRADE SYMBOL | DIAGNOSTIC FEATURES |
|---|---|---|
| Fresh | 1 | 100 per cent rock; no discolouration, decomposition, or other change. |
| Slightly Weathered | 2 | 100 per cent rock; discontinuity surfaces discoloured; the rock material may be discoloured. |
| Moderately Weathered | 3 | Soil resulting from decomposition of the rock forms up to 50 percent of the mass. |
| Highly Weathered | 4 | Soil resulting from decomposition of the rock forms between 50 and 100 per cent of the mass. |
| Completely Weathered | 5 | The rock is changed to a soil in which the original rock texture is (mainly) preserved. |
| Residual Soil | 6 | The rock is completely changed to a soil in which the original rock texture has been completely destroyed. |

CORE STONES AND LITHORELICS SHOULD BE REFERRED TO AS BOULDERS, COBBLES, OR GRAVEL, OR THEIR SIZE STATED IN COMMENTS COLUMN

## TABLE 2 - CHEMICAL ALTERATION

| TERM | GRADE SYMBOL | DIAGNOSTIC FEATURES |
|---|---|---|
| Slightly Altered | 1 | Less than 5 per cent of rock mass shows evidence of alteration. |
| Moderately Altered | 2 | Five to 25 per cent of rock mass altered. |
| Strongly Altered | 3 | Twenty Five to 50 per cent of mass altered. |
| Highly Altered | 4 | Fifty to 100 per cent of mass altered. |

## TABLE 3 - STRENGTH

| SYMBOL | DESCRIPTION | FIELD IDENTIFICATION |
|---|---|---|
| S1 | Very soft soil or rock | Easily moulded by fingers; shows distinct heel marks |
| S2 | Soft soil or rock | Moulded with strong pressure from fingers; shows faint heel marks |
| S3 | Firm soil or extremely weak rock | Very difficult to mould with fingers, indented with fingernail; difficult to cut with hand spade |
| S4/R1 | Stiff soil or very weak rock | Cannot be moulded with fingers; crumbles under blow of hammer, or can be peeled with knife. |
| S5/R2 | Very stiff soil or weak rock | Very tough, difficult to move with hand pick; peeled by knife with difficulty, indented by blow of hammer. |
| R3 | Medium - strong rock | Cannot be scraped with knife; fractured with single blow of hammer. |
| R4 | Strong rock | Requires more than single blow of hammer to fracture. |
| R5 | Very strong rock | Requires many blows of hammer to fracture rock. |
| R6 | Extremely strong rock | Can only be chipped by repeated blows of hammer. |

SITE ______ NORTHING: ______ INCLINATION: ______ DATE: BOREHOLE: ______

STARTED ______

LOCATION ______ EASTING: ______ BEARING: ______ COMPLETION: ______ PAGE ___ OF ___

ELEVATION: ______

DRILLING METHOD ______ DRILLING CONTRACTOR: ______ LOGGED BY: ______

| DEPTH | WEATH ALT | ROCK TYPE | RQD | TOTAL CORE RECOVERY | DIP OF FRACTURES | | | | | | | | | | | | | | | | STRENGTH | | | COMMENTS |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | 90-70 | | | | 70-50 | | | | 50-30 | | | | 30-0 | | | | | | | |
| | | | | | NO | ROUGH | ALTER | TYPE | NO | ROUGH | ALTER | TYPE | NO | ROUGH | ALTER | TYPE | NO | ROUGH | ALTER | TYPE | | | | |
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# APPENDIX E

## Standard Procedure for Collection of Geological Data at Thompson Mine

| | |
|---|---|
| E1 to E14 | Mines Exploration Standards for the Collection and Documentation of Geotechnical Data & Figures 2 to 4 |
| E15 | Addendum 1 - Simplified line survey procedure |
| E16 to E23 | Addendum 2 - Option D - Rock Mass Quality Tables |

# MINES EXPLORATION

# STANDARDS FOR THE COLLECTION AND DOCUMENTATION OF GEOTECHNICAL DATA

The need for the supply of geotechnical information has increased dramatically with the advent of VBM. Data supplied by Mines Exploration to Mine Engineering is more than ever having an impact on mine design and behaviour predictability. Data must be collected and evaluated based on set procedures such that methods are consistent and results factual.

These procedures are meant to be supplementary to current departmental standards and pertain exclusively to geotechnical requirements.

## Required Results

The collection and evaluation of geotechnical data must result in:-

1. A visual representation of structural defects on plans and sections.
2. An awareness of major structures which may result in structural instability.
3. An evaluation of potential structural modes of failure.
4. Documentation of rock mass characteristics.
5. The data being supplied to the user in an acceptable format and it being used both in the development of a mining plan and in predicting behaviour.

Procedures requiring further standardization include:-

1. Mapping.
2. Recording of geotechnical data.
3. Line surveys.
4. Line survey evaluation.
5. Rock mass quality estimations.
6. Stope and structural stability analyses.

## 1. Mapping

All mine excavations must be mapped. Structural data should be recorded such that a continuous pictorial representation of weakness plane characteristics is evident. It must be remembered during mapping that structural data recorded may be used in a detailed stability analysis which will only be as accurate as the mapped data. All structural orientations should be accurately plotted so that when measured the plotted plane will give an accurate azimuth. All major shears or joints must be recorded when evident that they are unique structures. It is sufficient to record measured joint set characteristics, strikes and dips etc. every 20-40 feet (20 feet across strike, 40 feet along strike). Single joints <3' in length should be excluded.

Note: A unique structure is a weakness plane which is not a common characteristic of the overall rock mass (an anomaly).

Care should be taken to record the exposed length of a significant single structures. It is important to know if a structure will influence both walls. Additional comments on infilling types and thickness, joint frequency, ore contact characteristics would aid in stability studies and supplement line survey data. The ore and wall rock types must be clearly recorded as lithological changes will influence estimated rock mass qualities, stability and anticipated wall blast damage. Avoid descriptive terms such as schistose quartzite or highly siliceous schist. These terms can be misinterpreted. The rock should be named first and followed by a description.

Make it routine to note general rock characteristics, to attempt visual estimates of RQD and rock quality and to look for and note potential causes of instability.

* Think about how you believe the stope or block will behave during mining.

## 2. Recording of Geotechnical Data

Geotechnical data which must be recorded on geological sections and plans and ultimately represented on layout drawings has four general sources:

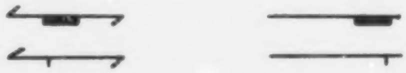
1. 20 Scale mapping.
2. Line surveys.
3. Borehole RQD values.
4. PD hole photography.

1. Mapped data should be plotted on both sections and plans noting major structures, joint sets etc. and must be factually represented on layout drawings. (refer to mapping standards).

2. Line survey data should be plotted on 20 scale plans. Only major structures (shears, major single joints) should be plotted as well as a representation of joint set data (typical attitude of each joint set). It is sufficient to plot the planes defining the poles of joint sets as determined on stereonet. Plots of weakness planes representing line survey results should use a block style dip symbol to allow differentiation from actual routine mapping data.

   e.g. line survey

   routine mapping



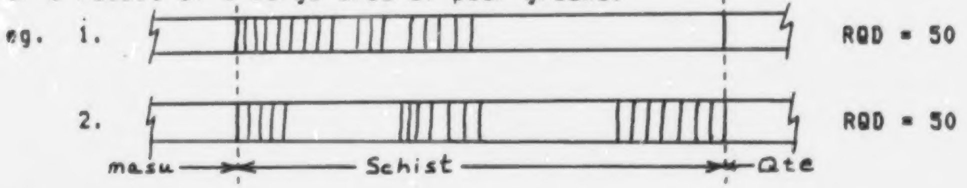
   The beginning and end points of a line survey should be noted on plan with a reference number such that the filed line survey detail can be referred to. (Refer to Sterinco line survey nomenclature).

3. RQD data collected should be represented on sections and plans as a visual record of relevant rock quality.

4. Photographs of drill holes (ore zone plus 20 feet either side of ore zone) should be reviewed such that down dip or lateral continuity of

shear zones or poor quality ground can be established. Such weakness zones should also be represented on sections and plans. This is an important source of information as drill holes may define structures not identified through mapping and line surveys and also may aid in better defining the attitude of structures which were encountered in mine openings.

All of this data should be recorded on the log. However, photographs give a better feel for logged characteristics, aid in relating visual estimations of RQD during mapping and aid in determining whether RQD values recorded for a rock unit are uniform or a result of a large area of poor ground.



## 3. Line Survey

The purpose of a line survey is to collect a sample of the weakness plane characteristics of a rock mass which is interpreted to represent the overall structural characteristics of the rock mass. Such planes strongly influence rock mass strength and the response of the rock mass to mining. For example, if a rock mass consists of a very strong intact rock type with few planes of weakness, it may exhibit brittle failure characteristics. If mining induces stresses exceed the strength of this type of rock mass, failure may be violent. On the other hand if the same rock mass is penetrated by a large number of continuous weakness planes (e.g. foliation, bedding) it will behave in a very different manner. Not only will the rock mass be weaker but excess stresses may be dissipated by movement along the various planes of weakness. Rather than violent breakage occurring, large deformations or slippage will be experienced. It is this second style of behaviour that describes the way in which the immediate hangingwall and footwall rocks at Thompson Mine behave.

The quality of a rock is related to the attitude of weakness planes, the length and frequency of weakness planes, the quality of the weakness plane as defined by infilling type, and thickness, the fluid conductivity and stress. Refer to Addendum 1, line survey method for this standard.

One line survey should be completed on each of the top and bottom sublevels of a VBM workplace. Ideally, data should be collected parallel to as well as at right angles to strike. The results of line surveys completed on top and bottom sublevels should be compared and if results are similar, no further line surveys are considered necessary. However, if results differ, additional surveys would be warranted. Each line survey should include approximately 50 measurements. All line survey data should be recorded in English (not metric).

The specific location for completion of a line survey should be picked at random. ie. Do not pick a location because of the known occurrences of specific structures.

Line surveys should not include single joints with a length of <3 feet. These structures are considered to play a minor role in structurally controlled instability.

Joint set characteristics or the characteristics of a continuous weakness plane should be recorded at minimum every 10 feet.

## 4. Line Survey Evaluation

Line surveys are the primary source of data for estimation of rock quality. All methods of estimating rock quality (eg. NGI, Geomechanics Classification etc.) generally use the same data input.

Using the sort and distribution functions on Lotus is, to date, the simplest method of evaluating line surveys data. This simply involves inputting the line survey data sheet on the PC and sorting the data by dip direction to determine joint set characterisitics and the distribution (modal values) for each input parameter. (Refer to figures 2 and 3).

All line survey weakness planes should be plotted on a stereonet as this will determine the number of joint sets, joint set poles, major structures (shears etc) and will aid in completion of stability analyses. (Refer to Figure 4).

It is useful to complement line survey data plotted on a stereonet with data picked up during routine mapping. This will add to the database, will confirm the line survey as being representative of joint set orientations, give more accurate pole determinations (avg. weakness plane orientation) and also include major structures which were not identified in the line survey. A stereo plot of weakness planes should be divided into shears, joint sets and major single joints. Major structures will be studied individually in stability analyses.

Inco is currently experimenting with a simplified line survey procedure. (Refer to Addendum 1)

## 5. Rock Mass Quality

The primary method currently used for the calculation of Rock Mass Quality is the NGI (Q) system. The calculation has the form:-

$$Q = \frac{RQD}{Jn} \times \frac{Jr}{Ja} \times \frac{Jw}{SRF}$$

where Q = rock quality designation

Jn = joint set number - related to number of joint sets

Jr = joint roughness number - related to weakness plane roughness

Ja = joint alteration number - related to the type - quality of infilling

Jw = joint water number = relates to weakness plane fluid conductivity

SRF = stress reduction factor

Refer to Addendum 2 - Rock Mass Quality tables.

The calculation RQD/$_{Jn}$ represents the relative size of blocks bounded by weakness planes. The calculation Jr/$_{Ja}$ represents the relative ability for movement to occur along weakness planes. The calculation Jw/$_{SRF}$ represents the hydrological and stress related influences on the quality of the rock.

Diamond drill holes are the best source for RQD data. Average RQD values should be calculated for each of FW, ore zone and HW for production blocks, but only for the first 20 feet of wall rock from the ore contact.

All other input parameters for the calculation of rock quality can be retrieved from line survey results or from borehole logs. The Jn and Jr values are best retrieved from line survey results. Ja values are more accurately recorded in borehole log descriptions.

A table should be completed for each workplace which documents the input values for the HW, FW, and ore zone of each block. Refer to Table.

Results of the NGI (Q) System calculation of rock quality can be compared to the results obtained from the Geomechanics Classification System for estimating quality.

**Note:** The Mines Exploration Borehole System has the capacity to calculate rock quality values. All rock quality parameters are retrieved from borehole log descriptions for use in calculating Q values. Rock Quality can be calculated for a requested borehole interval. As well, if line surveys have defined specific rock quality parameters these can be input for use calculation of new rock quality values. A judgement must be made in using this program on whether to rely totally on recorded borehole data or to input new parameters based on additional information from line surveys.

## ROCK MASS QUALITY DATA

Level ____________________________ Workplace __________________

| Block | (Hw,Ore,Fw) Location | RQD | Jn | Ja | Q' | Jw | SRF | Q |
|---|---|---|---|---|---|---|---|---|

## Structural Stability Analyses

It is insufficient to record structural data and estimated rock quality without investigating in detail the potential role that major structures or the intersections of weakness planes will have on the stability of mine openings. An example is the possibility of a stope having rocks of excellent quality but there being a major fault that passes through a block and which renders all other factors meaningless.

The first step in structural stability studies is to isolate all unique and major weakness planes and define them on a steronet treating each structure individually. Next plot the planes which represent the poles of the dominant joint sets.

The types of structurally controlled wall failure which will be encountered underground include sliding, toppling and wedge type failure. Back failure types include block drop out or wedging. Seldom is wall or back failure a result of the occurence of one weakness plane. Failure generally occurs as a result of the intersection of two or more weakness planes.

Potential wall failure as a result of the intersection of structures can easily be defined via stereonet. This involves plotting the various planes of weakness (actual shears, major joints or joint set poles) and also the plane representing the wall being studied. This will define potential daylighting and in the case if wedge failures indicate the azimuth and plunge of the line of intersection between two planes. Remember that the friction angle is approximately 30 degrees at Thompson Mine. Planes or plane intersections which have a dip of less than 30 degrees should be interpreted as being stable.

## Mine Engineering - Mines Exploration Liaison

Geotechnical Data collection and evaluation is futile if data is not supplied to Mines Engineering in a useable format. It is considered factual and is used in earnest by Mine Engineering when determining a mining plan and predicting stope behaviour.

Line survey results, rock quality parameter data, plans and sections and estimates of potential structural instability should be reviewed in detail with Mine Engineering, should be documented in a geotechnical report to Engineering and properly filed for reference in the Mines Exploration office.

# BASE DATA

1
STOPE 1000 LEVEL

| NORTH | EAST |
|---|---|
| 30152 | 9897 |
| 30160 | 9846 |

BASED.

| NUMBER | DISTANCE | TYPE | DIP DIR. | DIP ANGL. | LENGTH | END | ROUGHN | INFL | APP. WIDTH | FL. HARDN | WATER | WAV. | LITHO. | HARD. | (SPACING) COMMENTS |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 6 | JS | 175 | 90 | 2 | 0 | 4 | | VT | | D | P | SCH | | 1.5' SPACING |
| 2 | 7 | JS | 100 | 60 | 2 | 0 | 5 | | VT | | D | C | SCH | | 3"-1' |
| 3 | 7 | JS | 350 | 75 | 1 | 0 | 1 | | O | | D | P | SCH | | 5"-7" |
| 4 | 9 | J | 75 | 35 | 8 | 1 | 3 | | VT | | D | P | SCH | | |
| 5 | 9 | JS | 80 | 45 | 8 | 1 | 2 | | VT | | D | P | SCH | | 10" |
| 6 | 11 | J | 0 | 85 | 5 | 1 | 5 | | O | | D | C | SCH | | |
| 7 | 14 | JS | 95 | 50 | 6 | 0 | 5 | | PO | | D | P | QTE | | 2"-5" |
| 8 | 15 | J | 275 | 30 | 4 | 0 | 1 | CARB | T | | D | P | QTE | | |
| 9 | 15 | JS | 90 | 60 | 2_10 | 1 | 1 | | VT-O | | D | P | QTE | | 1.5-4" ALONG BEDDING |
| 10 | 18 | J | 100 | 65 | 2 | 0 | 5 | MICA | T | | D | P | QTE | | |
| 11 | 16 | JS | 190 | 85 | 5 | 1 | 7 | | T | | D | P | QTE | | 2.5-3" |
| 12 | 20 | JS | 235 | 80 | 3 | 0 | 1 | | T | | D | C | QTE | | 1.5" |
| 13 | 21 | JS | 60 | 80 | 3 | 0 | 1 | CARB | T | | D | C | QTE | | 0.5" |
| 14 | 22 | JS | 285 | 5 | 3 | 0 | 6 | MINOR SULF | T | | D | C | QTE | | 9" |
| 15 | 23 | JS | 110 | 50 | 2 | 0 | 3 | | VT | | D | P | QTE | | 2.5" |
| 16 | 25 | JS | 0 | 65 | 3 | 0 | 7 | | O | | D | P | QTE | | 6" |
| 17 | 25 | J | 295 | 15 | 1 | 0 | 7 | | VT | | D | P | QTE | | |
| 18 | 29 | J | 10 | 65 | 3 | 0 | 3 | | T | | D | U | QTE | | |
| 19 | 37 | JS | 120 | 55 | 12 | 2 | 7 | | VT | | D | P-C | QTE | | 3"-7" ALONG BEDDING |
| 20 | 37 | JS | 205 | 70 | 1 | 0 | 2 | | VT | | D | P | QTE | | 1"-6" |
| 21 | 42 | J | 355 | 85 | 6 | 0 | 3 | | VT | | D | P | QTE | | |
| 22 | 42 | JS | 275 | 15 | 1 | 0 | 7 | CARB | T | | D | P | QTE | | 3" |
| 23 | 43 | JS | 100 | 50 | 8 | 1 | 4 | CARB | T | | D | P | QTE | | 7" |
| 24 | 46 | J | 300 | 20 | 3 | 0 | 7 | CARB | T | | D | P | QTE | | |
| 25 | 51 | JS | 105 | 50 | 6 | 0 | 1 | | VT | | D | P | SCH | | 0.5"-1' |
| 26 | 51 | JS | 275 | 10 | 1 | 0 | 3 | CARB | T | | D | P | SCH | | 7" |
| 27 | 53 | JS | 115 | 85 | 0.5- | 0 | 5 | | VT | | D | F | QTE | | 1" |
| 28 | 53 | JS | 115 | 50 | 9 | 1 | 2 | CARB | T | | D | P | QTE | | 4" |
| 29 | 54 | JS | 20 | 75 | 2 | 0 | 2 | | VT | | D | P | QTE | | 2"-3" |
| 30 | 55 | J | 215 | 65 | 2 | 0 | 5 | | VT | | D | P | QTE | | |
| 31 | 56 | JS | 25 | 75 | 2 | 1 | 3 | | VT | | D | P | QTE | | 4" |
| 32 | 57 | J | 35 | 40 | 3 | 0 | 1 | | VT | | D | F | QTE | | |

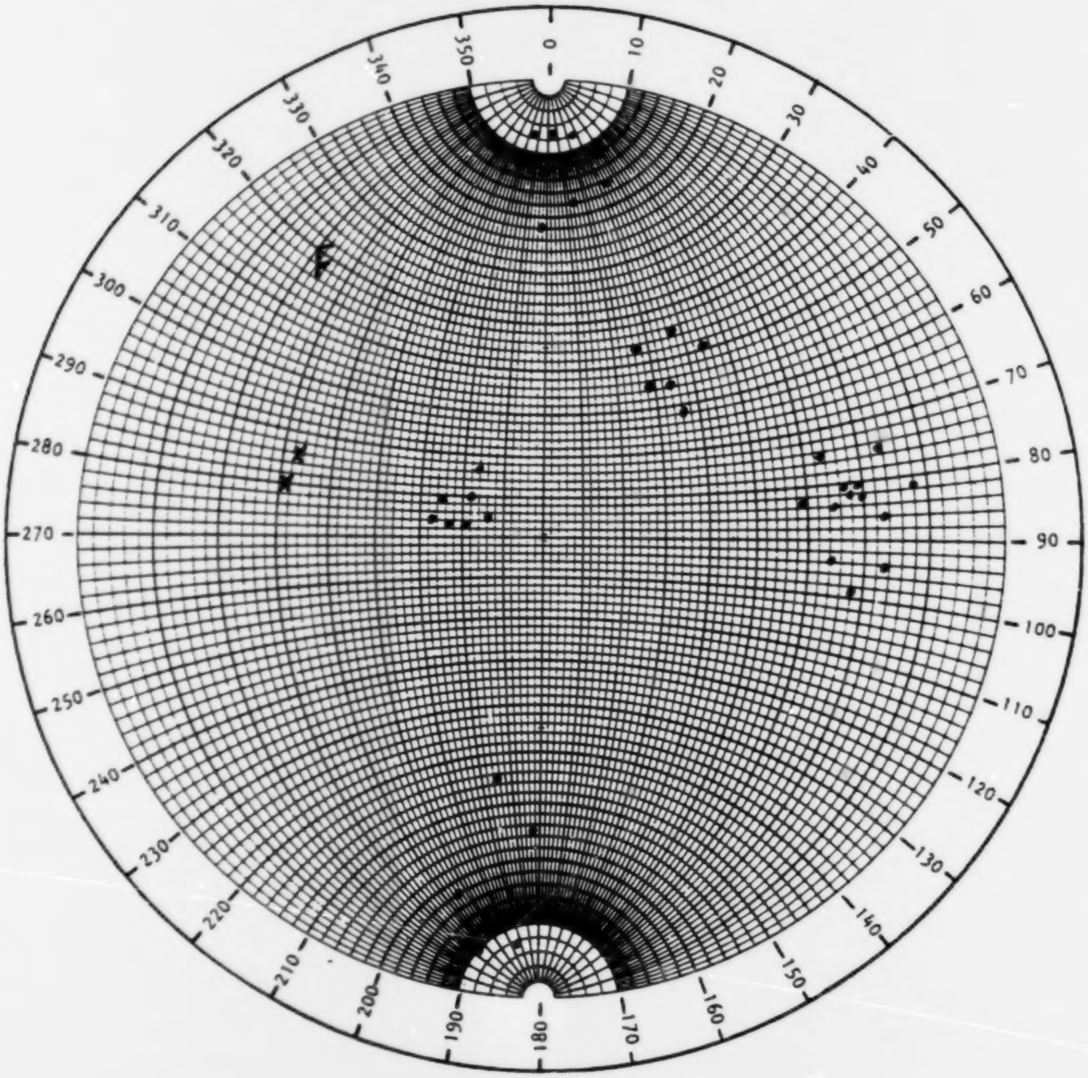
Figure 2

# SORTED DATA

| 1000 LEVEL | NUMBER | DISTANCE | TYPE | DIP DIR. | DIP ANGL. | LENGTH | END | ROUGHN | INFL | APP. WIDTH | FL. HARDN | WATER | WAV. | LITHD. | HARD. | (SPACING) COMMENTS |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| EAST | 6 | 11 | J | 0 | 85 | 5 | 1 | 5 | | 0 | | D | C | SCH | | |
| 9597 | 16 | 25 | JS | 0 | 65 | 3 | 0 | 7 | | 0 | | D | P | QTE | | 6" |
| 9646 | 18 | 24 | J | 10 | 65 | 3 | 0 | 3 | | T | | D | U | QTE | | |
| | 29 | 54 | JS | 20 | 75 | 2 | 0 | 2 | | VT | | D | P | QTE | | 2"-3" |
| SET 2 | 31 | 56 | JS | 25 | 75 | 2 | 1 | 3 | | VT | | D | P | QTE | | 4" |
| | 32 | 57 | J | 35 | 40 | 3 | 0 | 1 | | VT | | D | P | QTE | | |
| | 13 | 21 | JS | 60 | 80 | 3 | 0 | 1 | CARB | T | | D | C | QTE | | 0.5" |
| | 4 | 9 | J | 75 | 35 | 8 | 1 | 3 | | VT | | D | P | SCH | | |
| | 5 | 9 | JS | 80 | 45 | 8 | 1 | 2 | | VT | | D | P | SCH | | 10" |
| | 9 | 15 | JS | 90 | 60 | 2-10 | 1 | 1 | | VT-O | | D | P | QTE | | 1.5-4" ALONG BEDDING |
| | 7 | 14 | JS | 95 | 50 | 6 | 0 | 5 | | PO | | D | P | QTE | | 2"-5" |
| | 23 | 43 | JS | 100 | 50 | 8 | 1 | 4 | CARB | T | | D | P | QTE | | 7" |
| | 2 | 7 | JS | 100 | 60 | 2 | 0 | 5 | | VT | | D | C | SCH | | 3"-1" |
| | 10 | 16 | J | 100 | 65 | 2 | 0 | 5 | MICA | T | | D | P | QTE | | |
| | 25 | 51 | JS | 105 | 50 | 6 | 0 | 1 | | VT | | D | P | SCH | | 0.5"-1" |
| SET 1 | 15 | 23 | JS | 110 | 50 | 2 | 0 | 3 | | VT | | D | P | QTE | | 2.5" |
| | 27 | 53 | JS | 115 | 85 | 0.5 | 0 | 5 | | VT | | D | P | QTE | | 1" |
| | 28 | 53 | JS | 115 | 50 | 9 | 1 | 2 | CARB | T | | D | P | QTE | | 4" |
| | 19 | 37 | JS | 120 | 55 | 12 | 2 | 7 | | VT | | D | P-C | QTE | | 3"-7" ALONG BEDDING |
| | 1 | 6 | JS | 175 | 90 | 2 | 0 | 4 | | VT | | D | P | SCH | | 1.5" SPACING |
| SET 2 | 11 | 18 | JS | 190 | 85 | 5 | 1 | 7 | | T | | D | P | QTE | | 2.5-3" |
| | 20 | 37 | JS | 205 | 70 | 1 | 0 | 2 | | VT | | D | P | QTE | | 1"-6" |
| | 30 | 55 | J | 215 | 65 | 2 | 0 | 5 | | VT | | D | P | QTE | | |
| | 12 | 20 | JS | 235 | 80 | 3 | 0 | 1 | | T | | D | C | QTE | | 1.5" |
| | 26 | 51 | JS | 275 | 10 | 1 | 0 | 3 | CARB | T | | D | P | SCH | | 7" |
| | 22 | 42 | JS | 275 | 15 | 1 | 0 | 7 | CARB | T | | D | P | QTE | | 3" |
| SET 4 | 8 | 15 | J | 275 | 30 | 4 | 0 | 1 | CARB | T | | D | P | QTE | | |
| | 14 | 22 | JS | 285 | 5 | 3 | 0 | 6 | MINOR SULF | T | | D | C | QTE | | 9" |
| | 17 | 25 | J | 295 | 15 | 1 | 0 | 7 | | VT | | D | P | QTE | | |
| | 24 | 46 | J | 300 | 20 | 3 | 0 | 7 | CARB | T | | D | P | QTE | | |
| | 3 | 7 | JS | 350 | 75 | 1 | 0 | 1 | | 0 | | D | P | SCH | | 5"-7" |
| SET 2 | 21 | 42 | J | 355 | 85 | 6 | 0 | 3 | | VT | | D | P | QTE | | |

Figure 3

FIGURE 4.



**JOINT SETS**
(Estimate Poles)

| | DIP DIR. | DIP |
|---|---|---|
| 1. | 85° | 70° |
| 2. | 5°&185° | 78° |
| 3. | 35° | 50° |
| 4. | 282° | 20° |

**FAULT**
320/75

**LEGEND**

F — SHEAR
• — JOINTS
x — MAJOR JOINTS

## Addendum - 1

EQUIPMENT REQUIRED: ANGLE RULE, ORIENTING DEVICE ( DOUGLAS PROTRACTOR WITH ORIENTING ARM), 30M MEASURING TAPE, MARKER CRAYON, PAINT, RECORDING SHEETS.

### METHOD

1) Select survey site - choose clear exposure.

2) Anchor tape and set line 10 to 30 M long ~ chest height.

3) Record survey limits on wall and on plan.

4) Identify 1st structure nearest tape anchor and within a 0.5M window above and below tape.

5) Identify as set #1 and record dip & dip direction.

6) Mark structure with crayon - proceed along tape counting and marking all structures with similar orientation within the scan window. Record frequency.

7) Identify additional structural parameters in comments column.

ROCK TYPE - Take samples if unsure
WATER - Considered dry unless otherwise noted.
TYPE OF STRUCTURE - Shear, fault, contact,
 - Assume joint unless otherwise noted.
INFILLING - Mud, mineral, graphite, calcite
 - Assume tight joint unless otherwise noted
ROUGHNESS - Use carpenters comb
 - Assume mid-range (5-6-7) unless otherwise noted

Addendum - 2

## OPTION D - ROCK MASS QUALITY

### INTRODUCTION

Option D displays Q values through borehole log description scan, conversion for, borehole log entrv number standards to Q value parameter numbers and calculation of Q values via the formula:

$$Q = \frac{RQD \times Jr \times Jw}{Jn \qquad Ja \qquad SRF}$$

The program will go to default values (mine average values shown below) after conversion from log entry numbers to Q parameter numbers under the following conditions:

1. RQD has no default value
2. when Jn < 9
3. only if Jr does not exist in log description
4. only if Ja does not exist in log description
5. Jw and SRF always default to 1

Default values are:

Jn = 9
Jr = 1.5
Ja = 1.0
Jw = 1
SRF= 1

Under normal conditions Jn will be 9 or a higher number more than 3 joint sets are identified as a result of line surveys and the Jr value default should be changed based on line survey results. Ja is more well defined in log descriptions and normally will not be changed. Jw and SRF default values of 1 can be changed based on conditions identified through mapping on line surveys. RQD is always retrieved from log descriptions.

## LINE SURVEY THOMPSON MINE

LEVEL ______________ DATE ___________

WORKPLACE ____________

LINE ORIENTATION ____________ RECORDED BY ___________

LINE LENGTH _________

| SET NO. | DIP | DIP DIR | FREQUENCY | COMMENTS |
|---|---|---|---|---|
| 1 | | | | |
| 2 | | | | |
| 3 | | | | |
| 4 | | | | |
| 5 | | | | |
| 6 | | | | |
| 7 | | | | |
| 8 | | | | |
| 9 | | | | |
| 10 | | | | |
| 11 | | | | |
| 12 | | | | |
| 13 | | | | |
| 14 | | | | |
| 15 | | | | |
| 16 | | | | |
| 17 | | | | |
| 18 | | | | |
| 19 | | | | |
| 20 | | | | |

```
 BHRTRM40                    ONLINE BOREHOLE SYSTEM                       87-03-
 BHRTRP40             BHS     BOREHOLE RETRIEVAL     BHS                  12:39:
*************************  ROCK MASS QUALITY ************************************
SELECT BOREHOLE :   73284 0
 INTERVAL FROM  : 25.0___
         TO     : 55.8___                 ( TO = 0 INDICATES 'TO THE END OF BH
DEFAULT VALUES
--------------
JN : 9_
JR : 1.5
JA : 1.00
JW : 1.00
SRF: 1.0
PF-KEYS ------------------------------------------------------------------- A
 1:HELP  3:END
```

```
BHRTRP40            ROCK MASS QUALITY                                      87-03-19
BOREHOLE    732840                        INTERVAL FROM:      25.0 TO      55.8
 DEPTH  LENGTH   ROCK #RQD #JN #JR    #JA   #JW   #SRF  #Q
------- ------- ---- ---- --- ---- ----- ----- ---- ----
   25.0                           *** INTERVAL START ***
   31.1     6.1 SCH    57   9  1.0  0.75  1.00  1.0    8
   37.6     6.5 PEG    67   9  1.0  0.75  1.00  1.0    9
   42.5     4.9 SCH    74   9  1.0  0.75  1.00  1.0   10
   45.6     3.1 PEG    78   9  3.0  0.75  1.00  1.0   34
   55.8    10.2 QTE    71   9  1.0  0.75  1.00  1.0   10
        -------      -----  *** INTERVAL END    ***  -----
AVERAGE    30.8       68.6                              11.8
  135.0 *****END OF BOREHOLE
 BHRTRM40                    ONLINE BOREHOLE SYSTEM                          87-03-
 BHRTRP40            BHS      BOREHOLE RETRIEVAL     BHS                     12:39:
************************* ROCK MASS QUALITY ********************************
SELECT BOREHOLE :  73284 0
 INTERVAL FROM  : 0.0
          TO    : 0.0                     ( TO = 0 INDICATES 'TO THE END OF BH
DEFAULT VALUES
--------------
JN : 12
JR : 1.5
JA : 1.00
JW : 1.00
SRF: 1.0
PF-KEYS ----------------------------------------------------------------- A
 1:HELP  3:END
```

```
BHRTRP40           ROCK MASS QUALITY                                    87-03-19
BOREHOLE    732840                         INTERVAL FROM:       0.0 TO 99999.9
 DEPTH  LENGTH  ROCK #RQD #JN #JR    #JA    #JW   #SRF   #Q
------- ------- ---- ---- --- ----  -----  -----  ----  ----
   0.0                          *** INTERVAL  START ***
  20.8     20.8 QTE    21  12  1.5  0.75   1.00   1.0      3
  25.0      4.2 IF     82  12  1.5  0.75   1.00   1.0     13
  31.1      6.1 SCH    57  12  1.0  0.75   1.00   1.0      6
  37.6      6.5 PEG    67  12  1.0  0.75   1.00   1.0      7
  42.5      4.9 SCH    74  12  1.0  0.75   1.00   1.0      8
  45.6      3.1 PEG    78  12  3.0  0.75   1.00   1.0     26
  55.8     10.2 QTE    71  12  1.0  0.75   1.00   1.0      7
  61.0      5.2        71  12  1.0  0.75   1.00   1.0      7
  63.2      2.2 SCH    58  12  1.5  2.00   1.00   1.0      3
  68.8      5.6 SHR    13  12  1.5  0.75   1.00   1.0      2
  71.2      2.4 SCH     0  12  3.0  2.00   1.00   1.0      0
  99.8     28.6 SCH    34  12  1.0  0.75   1.00   1.0      3
 135.0     35.2        64  12  1.5  0.75   1.00   1.0     10
        -------      -----  *** INTERVAL  END    ***    -----
AVERAGE   135.0        49.5                               6.5
 135.0 *****END OF BOREHOLE
```

```
      BHAPRT10                                          BOREHOLE RECORD                               DATE PROCESSED - MAR 03/86
                                                        ***************
*************************************************************************************************************************************
  DEPTH  LENGTH  SAMPLE  EST.G    CU    NI      ZN     S.G.    S    CU   CRE ROCK              DESCRIPTION                 PROGRESSIVE TOTALS
                                                                                                                                CU        NI
   69.6    0.8   595432   1.0   0.08  3.97                               R1,JN4,JR2,JA0,RQD 013
   71.2    1.6   595433    .1   0.06  0.25                     0.06 MSCH      10% FE-MG PO PN, IN SCH BIOT, LCLY PEGL  2.68      35.92
                                                              0.01       SCH  BIOT WITH BNDS PEG & QTZ, SOMEWHAT SHRD  2.77      36.32
                                                                              & ALTD, 75
   72.3    1.1                                                                R3,JN3,JR5,JA2,RQD 033
   99.8   27.5                                                           LC
                                                                         SCH  BIOT, MOSTLY SLCS, LCLY GRFR, NMRS BNDS
                                                                              CREAM-GY PEG, OCC AND WHT QTZ TO 0.1 FT
                                                                              SVRL BNDS QTZ MICS, HLY GRFR 92.7-94.0,
                                                                              P BND SKN? NEAR LWR CT, OCC CARB FILLED
                                                                              FRCT, MOSTLY SLMN?-RICH, 65-80
  135.0   35.2                                                                R3,JN4,JR1,JA0,RQD 034
                                                                         SKN  CARB QTZ OLVN, NMRS MICS TO SCHS BNDS
                                                                              TRGT FIRST 10.0 FT, GRDG INTO LT-MED
                                                                              GRN-GY DH, BCMG MORE MASS CARB-SLCT
                                                                              RICH DH WITH OLVN-RICH PATCHES, WHT QTZ
                                                                              111.2-112.0 & R OTHER SMLR BND, R TR
                                                                              SULP, 80-75-80
                                                                              R2,JN4,JR2,JA0,RQD 064
  LOGGED BY....DV BRAUN, EXT, INCO, 340.1 SUBLEVEL                            FOOT OF HOLE
                                                                                            GRID SYSTEM - THOMPSON
*************************************************************************************************************************************
```

E22

# NGI ROCK CLASSIFICATION SYSTEM
# PARAMETER EVALUATION

| Item and Description | Parameter | Value |
|---|---|---|
| **Number of Sets of Discontinuities** | $J_n$ | |
| Massive | | 0.5 |
| One set | | 2.0 |
| Two sets | | 4.0 |
| Three sets | | 9.0 |
| Four or more sets | | 15.0 |
| Crushed rock | | 20.0 |
| **Roughness of Discontinuities** | $J_r$ | |
| Noncontinuous joints | | 4.0* |
| Rough and wavy | | 3.0* |
| Smooth and wavy | | 2.0* |
| Rough and planar | | 1.5* |
| Smooth and planar | | 1.0* |
| Slick and planar | | 0.5* |
| Filled discontinuities | | 1.0* |
| **Filling and Wall-Rock Alteration, Essentially Unfilled** | $J_a$ | |
| Healed | | 0.75 |
| Staining only; no alteration | | 1.0 |
| Silty or sandy coatings | | 3.0 |
| Clay coatings | | 4.0 |
| **Filling and Wall-Rock Alteration, Filled** | $J_a$ | |
| Sand or crushed rock filling | | 4.0 |
| Stiff clay filling less than 5 mm thick | | 6.0 |
| Soft clay filling less than 5 mm thick | | 8.0 |
| Swelling clay filling less than 5 mm thick | | 12.0 |
| Stiff clay filling more than 5 mm thick | | 10.0 |
| Soft clay filling more than 5 mm thick | | 15.0 |
| Swelling clay filling more than 5 mm thick | | 20.0 |
| **Water Conditions** | $J_w$ | |
| Dry | | 1.0 |
| Medium water inflow | | 0.66 |
| Large inflow; unfilled joints | | 0.5 |
| Large inflow; filled joints with washout | | 0.33 |
| Large inflow; filled joints; high transient inflow | | 0.2 to 0.1 |
| Large inflow; filled joints; high continuous inflow | | 0.1 to 0.05 |
| **Stress Reduction Class** | SRF | |
| Loose rock with clay-filled discontinuities | | 10.0 |
| Loose rock with open discontinuities | | 5.0 |
| Shallow depth (50 m or less) rock with clay-filled discontinuities | | 2.5 |
| Rock with tight unfilled discontinuties; medium stress | | 1.0 |
| **Rock-Mass Quality for Tunneling** | Q | |
| Exceptionally poor | | Less than 0.01 |
| Extremely poor | | 0.01 to 0.10 |
| Very poor | | 0.10 to 1.00 |
| Poor | | 1.00 to 4.00 |
| Fair | | 4.00 to 10.0 |
| Good | | 10.0 to 40.0 |
| Very good | | 40.0 to 100.0 |
| Extremely good | | 100.0 to 400.0 |
| Exceptionally good | | 400.0 to 1000.0 |

Adapted from: Barton, N., Lien, R., and Lunde, J., 1974.

* Add 1.0 if the mean joint spacing exceeds 3.0 m.

# APPENDIX F

## Instructions for Running the Sterinco Stereographic Computer Listings

| | |
|---|---|
| F1 to F8 | Sterinco Program |
| F9 to F15 | Sterinco Program Listing |
| F16 to F25 | Block Program |
| F26 to F43 | Block Program Listing |

# STERINCO PROGRAM

## PROGRAM STERINCO

The following are the instructions for running the Sterinco Stereographic Projection Program. This program was developed for Inco by Golder Associates.

The program is on a double density diskette, available upon request, that can only be run on I.B.M. AT with high density drive unit and a 8037 math coprocessor. Usually this is drive "A" on I.B.M. AT's. Prior to using this program a copy of the diskette should be made and the original filed. This copy will be used to store your files and can be expected to hold about 20 line survey's before another copy is required.

1. a) To start the program boot up using DOS 3.0 or more recent.

   b) Load disk up on appropriate drive unit and initiate program by typing "STERINCO".

   c) The following screen will appear.

```
íëëëëëëëëëëëëëëëëëëëëëëëë¡
•      CANMET SERIES     °
•    INPUT STEREO DATA   °
°      Version 1.0       °
•      August 1989       °
åëëëëëëëëëëëëëëëëëëëëëëëëƒ




Press space bar to continue
```

d) Press the space bar and the menu below will be put up on the screen.

```
CANMET Series - STERINCO (vers. 1.0) - 09-12-1989 - 07:36:59
____________________________________________________________

                    M E N U

                1 - CREATE A NEW STEREO DATA FILE

                2 - APPEND TO AN EXISTING FILE

                3 - CONVERT INCO LOTUS 123 FILE

                4 - RETURN TO DOS

                    SELECT?
```

2. When you pick Option 1 or 2 you will be requested for a valid: drive, filename and extension.

   a) The drive is "A"
   b) The filename consists of a maximum of 8 characters with no spaces. The following guidelines are to be used to generate filenames:

      <u>FILENAME FOR STERINCO PROGRAM</u>

```
      _ : _ _  _ _ _   _ _ _  .  _ _ _

DRIVE LEVEL WRKPL. IDENT. SURVEY EXTENSION
```

DRIVE - is usually A drive
LEVEL - a two number designation only
WORKPLACE OR SECTION - a three number designation only
IDENT. - can use the next two spaces if required. Follow the guidelines below:

| | |
|---|---|
| ST | - STOPE |
| D | - DRIFT (THE PREVIOUS NUMBER SHOULD REFER TO SECTION NOT THE WORKPLACE) |
| R | - RAMP |
| B_ (1-9) | - BLOCK AND BLOCK NUMBER |
| S_ (1-9) | - INDICATES SUBLEVEL AND THE SUBLEVEL NUMBER. |

SURVEY - survey number (A-Z).

EXTENSION - always use "DAT" (for Data)

Examples:

A:34231S3B.DAT - A Drive, 3400 Level, 231 Workplace Sublevel 3, Second Survey.

A:28896DA.DAT - A Drive, 2800 Level, 896 Section, Survey was done in the Drift, First Survey.

NOTE: MAKE SURE NO SPACES ARE LEFT IN THE FILENAME!

3. The computer will now request the type of input to expect, traverse name (use either T-1 or T-3 depending on which headframe you are working), etc., etc., as show below.

```
CANMET Series - STERINCO (vers. 1.0) - 09-12-1989 - 07:53:05

   Enter Stereo Filename (b:xxxxxxxx.dat) : b:dave.dat

   Select Input Type:

          1 - Convert strike and plunge angles

          2 - Directly input dip and dip-direction

              Select? 1

   Enter Traverse Name (1-7 Characters) : T-1

   Data for Terzaghi Correction (y/n) ? y

   Enter Plunge of the Traverse Line (0-90) ? 0

   Enter Trend (Azimuth) of the Traverse Line (0-360) ? 0
```

4. Depending on the input option selected the computer will generate one of the following input screens.

```
CANMET Series - STERINCO (vers. 1.0) - 09-12-1989 - 07:51:09
____________________________________________________________________________

Number  Type  Strike  Plunge  Rock   |  Number  Type  Strike  Plunge  Rock

   1                                      16
   2                                      17
   3                                      18
   4                                      19
   5                                      20
   6                                      21
   7                                      22
   8                                      23
   9                                      24
  10                                      25
  11                                      26
  12                                      27
  13                                      28
  14                                      29
  15                                      30


                     Press <Esc> to terminate input
```

```
CANMET Series - STERINCO (vers. 1.0) - 09-12-1989 - 07:54:59
____________________________________________________________________________

Number  Type  Dip-Dir   Dip   Rock   |  Number  Type  Dip-Dir   Dip   Rock

   1                                      16
   2                                      17
   3                                      18
   4                                      19
   5                                      20
   6                                      21
   7                                      22
   8                                      23
   9                                      24
  10                                      25
  11                                      26
  12                                      27
  13                                      28
  14                                      29
  15                                      30


                     Press <Esc> to terminate input
```

The computer enters input mode. Within the input mode only the following keys are available:

a) All alphabetic and numeric keys. Use numeric keys located in the top row. All lower-case alphabetic keys are automatically converted to upper case letters.

b) Directional keys. The directional keys located on the numeric pad are used to move the highlighted cursor about the screen. A beep will sound if you try to move beyond the current window.

c) -PgUp- and -PgDn- keys. The -PgUp- and -PgDn- keys are used to refresh the screen for additional input or to review the data already entered.

d) The -"- key. This key represents a ditto mark and the last line entered will be duplicated each time the ditto key is entered.

e) The -Back Space- key. This key results in the character entered to be erased from the input string.

f) The -Enter- or -Return- key. This key results in the last string entered to be displayed in the present cursor location and the cursor moved to the next position.

g) The -Esc- key. The escape key terminates the input mode, converts data, and returns to the program menu.

In input mode the data is entered by positioning the cursor to the desired location and keying in the information. Each stroke of the key, other then directional, paging and ditto keys are displayed in the lower right-hand corner.

Basic input data consists of the following four types:

a) Surface Type - Two character identification code defining joint, fracture or bedding. Suggest "J1", "J2", "J3", for various types of joints. "F1", "F2", "F3" for various types of faults. "B1", "B2", "B3" for various types of bedding.

b) Trend Data - The trend data consists of four or five characters. First character is always "N". Characters 2 to 3 or 4 represent the offset angle. The last character is the directional code "E" for east or "W" for west.

c) Plunge Data - The plunge data consists of two digit plunge angle plus a single direction code. e.g. "75W", "60E". Note Only "E" or "W" is accepted.

d) Rock Type - Use any unique three-character identification to define various type of rock type.

STERINCO program converts trend and plunge angles to dip and dip-direction angles as follows:

a) Dip Angle:

DIP - plunge angle.

b) Dip Direction Angle:

(i) When trend direction is "W" and plunge direction is "W":

DIPD = 360 - trend angle - 90

(ii) When trend direction is "W" and plunge direction is "E":

DIPD = 360 - trend angle +90

(iii) When trend direction is "E" and plunge direction is "W":

DIPD = trend angle - 90

(iv) When trend direction is "E" and plunge direction is "E":

DIPD = trend angle + 90.

5. When all data is correctly entered hit the -ESC- key and your data will be filed on the disk. The program should return you to DOS.

6. To produce the Stereographic plot type in:

"STEREO"

The prompts and sample answers are listed below. Make sure the format specification entered is (10x, I2, I3, and "A" drive specified for all files. The output filename must be followed with the extension "out".

7. When the program is complete the following screen will appear:

```
STEREO OUTPUT ASSIGNED TO :  A:34-231.OUT

 STEREO INPUT ASSIGNED TO : a:34-231.DAT
```

```
                    PLOT FINISHED

     * * * *  P R O G R A M   F I N I S H E D  * * * *

        Execution Terminated:    0

A:\>
```

a) To print your file type in:

"Print - Filename.  OUT"

Default to the assigned printer and the following screen should appear.

```
A:\>print 34-231.out
Name of list device [PRN]:
Resident part of PRINT installed

   A:\34-231.out is currently being printed.

A:\>
```

8. Log off and remove disk when complete.

   Each headframe will be responsible for keeping an up to date directory listing for all Sterinco backup disks.

# STERINCO PROGRAM LISTING

## AUGUST 1989

```
1 GOTO 10
2 SCREEN 0, 1: WIDTH 80: CLS : COLOR 7, 1: CLS
  PRINT "CANMET Series - STERINCO (vers. 1.0) - " + DATE$ + " - " + TIME$
  COLOR 4, 1: PRINT STRING$(80, "_")
  COLOR 7, 1: RETURN
10 REM CANMET STERINCO (Version 1.0) - PROGRAM HEADER
40 KEY OFF: SCREEN 0, 1: COLOR 5, 0, 0: WIDTH 80: CLS :
60 COLOR 7, 1: LOCATE 9, 27, 0: PRINT CHR$(213) + STRING$(24, 205) +
CHR$(184)
70 LOCATE 10, 27, 0: PRINT CHR$(179) + "     CANMET SERIES     " + CHR$(179)
75 LOCATE 11, 27, 0: PRINT CHR$(179) + "   INPUT STEREO DATA   " +
CHR$(179)
80 LOCATE 12, 27, 0: PRINT CHR$(179) + "      Version 1.0      " + CHR$(179)
90 LOCATE 13, 27, 0: PRINT CHR$(179) + "      August 1989      " + CHR$(179)
100 LOCATE 14, 27, 0: PRINT CHR$(212) + STRING$(24, 205) + CHR$(190)
120 COLOR 7, 4, 0: LOCATE 25, 25, 0: PRINT " Press space bar to continue ";
130 REM
140 CMD$ = INKEY$
150 IF CMD$ = "" THEN GOTO 140
170 IF CMD$ = " " THEN GOTO 190
180 GOTO 130
190 GOSUB 2
210 DIM IPLUNG$(2000), ITREND$(2000), ISURF$(2000), IROCK$(2000), ILEN(4),
ITAB(4)
220 CLOSE : GOSUB 600: GOSUB 1500: GOSUB 6000
225 IF ISELECT = 3 THEN GOTO 800
230 IF ISELECT = 2 THEN GOTO 260
235 PRINT : PRINT TAB(5); : INPUT "Enter Traverse Name (1-7 Characters) : ",
TNAME$: IF TNAME$ = "" THEN GOTO 250
240 TTL$ = "": PTL$ = "": PRINT : PRINT TAB(5); : INPUT "Data for Terzaghi
Correction (y/n) "; AN$
245 IF AN$ = "Y" OR AN$ = "y" THEN GOTO 250 ELSE GOTO 260
250 PRINT : PRINT TAB(5); : INPUT "Enter Plunge of the Traverse Line (0-90) "; PTL
   IF PTL >= 0 AND PTL <= 90 THEN GOTO 255 ELSE PRINT "*** Invalid Plunge
Angle ***": GOTO 250
255 PRINT : PRINT TAB(5); : INPUT "Enter Trend (Azimuth) of the Traverse Line (0-
360) "; TTL
   IF TTL >= 0 AND TTL <= 360 THEN GOTO 260 ELSE PRINT "*** Invalid Trend
Angle ": GOTO 255
260 PTL$ = RIGHT$("   " + STR$(PTL), 3): TTL$ = RIGHT$("   " + STR$(TTL), 3)
265 PRINT #2, "T" + TNAME$; TAB(12); PTL$; TTL$
270 GOSUB 3460: GOSUB 6100: IF IOPT = 1 THEN GOSUB 1700
280 GOSUB 2: GOTO 220
500 'Check Filename
510 IF INSTR(X$, ".") = 0 THEN X$ = X$ + ".STE"
520 RETURN
600 REM =======================( DISPLAY MAIN MENU
)========================
610 XT = 15: YT = 4: LOCATE YT, XT
620 PRINT "M E N U"
630 XT = 10: YT = YT + 2: LOCATE YT, XT
640 PRINT " 1 - CREATE A NEW STEREO DATA FILE"
650 YT = YT + 2: LOCATE YT, XT
660 PRINT " 2 - APPEND TO AN EXISTING FILE"
```

```
670 YT = YT + 2: LOCATE YT, XT
675 PRINT " 3 - CONVERT INCO LOTUS 123 FILE"
680 YT = YT + 2: LOCATE YT, XT
685 PRINT " 4 - RETURN TO DOS"
690 XT = 15: YT = YT + 2: LOCATE YT, XT
700 PRINT "SELECT"; : INPUT ISELECT
705 IF ISELECT = 3 THEN RETURN
710 IF ISELECT = 4 THEN GOTO 9000
720 GOSUB 2: PRINT : PRINT TAB(5); : INPUT "Enter Stereo Filename
(b:xxxxxxxx.dat) : ", F2$: X$ = F2$: GOSUB 500: F2$ = X$
730 IF ISELECT = 1 THEN OPEN F2$ FOR OUTPUT AS #2
750 PRINT : PRINT TAB(5); "Select Input Type:"
760 PRINT : PRINT TAB(10); " 1 - Convert strike and plunge angles"
770 PRINT : PRINT TAB(10); " 2 - Directly input dip and dip-direction"
780 PRINT : PRINT TAB(15); "Select"; : INPUT IOPT
790 RETURN
800 REM ==========( CONVERT LOTUS 123 FORMAT TO STEREO FORMAT
)===============
810 GOSUB 2: PRINT : PRINT TAB(5);
   INPUT "Enter Lotus 123 Filename (b:xxxxxxxx.prn) : ", F1$
815 IF INSTR(F1$, ".") = 0 THEN F1$ = F1$ + ".PRN"
820 OPEN F1$ FOR INPUT AS #1
825 X = INSTR(F1$, "."): F2$ = LEFT$(F1$, X) + "STE"
   PRINT : PRINT TAB(5); "Stereo Filename is "; F2$
830 OPEN F2$ FOR OUTPUT AS #2
835 PRINT : PRINT TAB(5); : INPUT "Enter Traverse Name (1-7 Chars) : ", TNAME$
840 TTL$ = "": PTL$ = "": PRINT : PRINT TAB(5);
   INPUT "Data for Terzaghi Correction (y/n) "; AN$
845 IF AN$ = "Y" OR AN$ = "y" THEN GOTO 850 ELSE GOTO 860
850 PRINT : PRINT TAB(5); : INPUT "Enter Plunge of the Traverse Line (0-90) "; PTL
   IF PTL >= 0 AND PTL <= 90 THEN
   GOTO 855
   ELSE
   PRINT "*** Invalid Plunge Angle ***"
   GOTO 850
   END IF
855 PRINT : PRINT TAB(5); : INPUT "Enter Trend (Azimuth) of the Traverse Line (0-
360) "; TTL
   IF TTL >= 0 AND TTL <= 360 THEN
   GOTO 860
   ELSE
   PRINT "*** Invalid Trend Angle "
   GOTO 855
   END IF
860 PTL$ = RIGHT$("   " + STR$(PTL), 3): TTL$ = RIGHT$("   " + STR$(TTL), 3)
   PRINT #2, "T" + TNAME$; TAB(12); PTL$; TTL$
865 LINE INPUT #1, LBUF$: IF EOF(1) THEN GOSUB 2: GOTO 220
870 REM test for blank record
875 LB = LEN(LBUF$): NX = 0
   FOR NC = 1 TO LB
   IF MID$(LBUF$, NC, 1) <> " " THEN GOTO 885
880 NEXT NC: GOTO 865
885 REM =====================( TEST FOR NEW TRAVERSE
)======================
890 LL = LB - 9: OUTBUF$ = "": OUTBUF$ = "D" + LEFT$(TNAME$ + "        ", 7) +
MID$(LBUF$, 10, LL)
895 PRINT #2, OUTBUF$
900 GOTO 865
1000 REM =====( REDUCE STRIKE AND PLUNGE ANGLE TO DIP AND DIP-
DIRECTION )=====
1010 NS$ = LEFT$(STRIKE$, 1): LS = LEN(STRIKE$) - 2: SVAL =
VAL(MID$(STRIKE$, 2, LS)): EW$ = RIGHT$(STRIKE$, 1)
1015 LS = LEN(PLUNGE$) - 1: IF PLUNGE$ = "90" THEN LS = LS + 1
1020 DIP = VAL(LEFT$(PLUNGE$, LS)): PDIR$ = RIGHT$(PLUNGE$, 1)
1025 IF PLUNGE$ = "90" THEN DDIP = SVAL - 180
1030 IF EW$ = "W" AND PDIR$ = "W" THEN DDIP = 360 - SVAL - 90
1040 IF EW$ = "W" AND PDIR$ = "E" THEN DDIP = 360 - SVAL + 90
1050 IF EW$ = "E" AND PDIR$ = "E" THEN DDIP = 0 + SVAL + 90
1060 IF EW$ = "E" AND PDIR$ = "W" THEN DDIP = 0 + SVAL - 90
1070 IF DDIP < 0 THEN DDIP = DDIP + 360
1080 IF DDIP > 360 THEN DDIP = DDIP - 360
1090 RETURN
1200 REM ========================( DISPLAY INPUT SCREEN
)==================
1210 GOSUB 2: LOCATE 4, 1
   IF IOPT = 1 THEN HEAD$ = "Number  Type  Strike  Plunge  Rock ": PRINT
HEAD$ + " | " + HEAD$
1215 IF IOPT = 2 THEN HEAD$ = "Number  Type  Dip-Dir   Dip   Rock ": PRINT
HEAD$ + " | " + HEAD$
1220 LOCATE 25, 25: PRINT "Press <Esc> to terminate input";
1300 REM =================( REFRESH DATA ENTRY DISPLAY SCREEN
)=================
1310 IP1 = IPS: IP2 = IPS + 15 - 1: X = 5
1320 FOR I = IP1 TO IP2
   X = X + 1: LOCATE X, 1
   PRINT RIGHT$("     " + STR$(I), 5);
   NEXT I
1330 IP1 = IPS + 15: IP2 = IP2 + 15: X = 5
1340 FOR I = IP1 TO IP2
   X = X + 1: LOCATE X, 41
   PRINT RIGHT$("     " + STR$(I), 5);
   NEXT I
1400 REM ========================( DISPLAY DATA TO SCREEN
)=================
1410 IF IMAX <= 0 THEN RETURN
1420 IP1 = IPS: IP2 = IPS + 15 - 1: X = 5: T2 = ITAB(2): T3 = ITAB(3)
   T4 = ITAB(4)
1430 FOR I = IP1 TO IP2
   IF I > IMAX THEN GOTO 1440
   X = X + 1
   LOCATE X, 10: PRINT ISURF$(I); TAB(T2); ITREND$(I); TAB(T3); IPLUNG$(I);
TAB(T4); IROCK$(I);
   NEXT I
1440 IP1 = IP1 + 15: IP2 = IP2 + 15: X = 5: T2 = T2 + 40: T3 = T3 + 40: T4 = T4 + 40
1450 FOR I = IP1 TO IP2
   IF I > IMAX THEN GOTO 1460
   X = X + 1
```

```
    LOCATE X, 50: PRINT ISURF$(I); TAB(T2); ITREND$(I); TAB(T3); IPLUNG$(I);
TAB(T4); IROCK$(I);
    NEXT I
1460 FG = 0: BG = 2: GOSUB 4800: RETURN
1500 REM ===================( ENTER PREVIOUS DATA
)===================
1510 IMAX = 0: IF ISELECT <> 2 THEN RETURN
1520 IF IOPT <> 1 THEN GOTO 1600
1530 X = INSTR(F2$, "."): F3$ = LEFT$(F2$, X) + "STE"
1540 OPEN F3$ FOR INPUT AS #3
1550 IF EOF(3) THEN GOTO 1590
1560 LINE INPUT #3, LBUF$: IMAX = IMAX + 1: ISURF$(IMAX) = LEFT$(LBUF$,
2): ITREND$(IMAX) = MID$(LBUF$, 3, 5)
1570 IPLUNG$(IMAX) = MID$(LBUF$, 8, 4): IROCK$(IMAX) = MID$(LBUF$, 12,
3)
1580 GOTO 1550
1590 CLOSE #1: OPEN F2$ FOR OUTPUT AS #2: RETURN
1600 REM re-enter previous dip and dip-direction data.
1610 OPEN F2$ FOR INPUT AS #3: LINE INPUT #3, TRAV$: TNAME$ =
MID$(TRAV$, 2, 7): PTL$ = MID$(TRAV$, 12, 3): TTL$ = MID$(TRAV$, 15, 3)
1620 IF EOF(3) THEN GOTO 1660
1630 LINE INPUT #3, LBUF$: IMAX = IMAX + 1: ISURF$(IMAX) = MID$(LBUF$,
12, 2): IPLUNG$(IMAX) = MID$(LBUF$, 14, 2)
1640 ITREND$(IMAX) = MID$(LBUF$, 16, 3): IROCK$(IMAX) = MID$(LBUF$, 19,
3)
1650 GOTO 1620
1660 CLOSE : OPEN F2$ FOR OUTPUT AS #2: RETURN
1700 REM ================( SAVE CURRENT STRIKE AND PLUNGE DATA
)==========
1710 X = INSTR(F2$, "."): F3$ = LEFT$(F2$, X) + "STE": OPEN F3$ FOR OUTPUT
AS #3
1720 FOR I = 1 TO IMAX
1730    LBUF$ = LEFT$(ISURF$(I) + "  ", 2)
1740    LBUF$ = LBUF$ + LEFT$(ITREND$(I) + "     ", 5)
1750    LBUF$ = LBUF$ + LEFT$(IPLUNG$(I) + "    ", 4)
1760    LBUF$ = LBUF$ + LEFT$(IROCK$(I) + "   ", 3)
1770    PRINT #3, LBUF$
1780 NEXT I
1790 CLOSE #3: RETURN
3460 REM =============( Data entry and special movement keys )===========
3470 ' row - current cursor row, col - current cursor column
3480 B$ = "": IPS = 1: IFIELD = 1: FREQ = 450: DUR = 10: COLADJ = 0: COL =
ITAB(IFIELD): NOCHR = ILEN(IFIELD): ROW = 6: GOSUB 1200: FG = 0: BG = 2:
GOSUB 4800
3490 LOCATE 23, 35: COLOR 0, 2: PRINT STRING$(40, 32): LOCATE 23, 36:
PRINT B$;
3495 COLOR 7, 0: X$ = INKEY$: IF X$ = "" THEN 3495
3500 'check special function key
3510 IF LEN(X$) = 2 THEN X$ = RIGHT$(X$, 1): X = ASC(X$): GOTO 3580
3520 X = ASC(X$): IF X = 34 THEN B$ = "": GOTO 5100
3530 IF X = 27 THEN B$ = "": CLS : RETURN
3535 IF X = 8 THEN LX = LEN(B$): B$ = LEFT$(B$, LX - 1): GOTO 3490
3540 IF X = 13 THEN GOTO 3750
3560 IF X > 47 AND X < 58 THEN GOTO 3740
```

```
3565 IF IOPT = 1 THEN 3570
3566 IF IFIELD = 2 THEN GOTO 3590
3567 IF IFIELD = 3 THEN GOTO 3590
3570 IF X > 64 AND X < 91 THEN GOTO 3740
3575 IF X > 96 AND X < 123 THEN X = X - 32: X$ = CHR$(X): GOTO 3740
3580 'direction keys
3590 IF X = 77 THEN GOTO 3750              'right
3600 IF X = 75 THEN GOTO 3800              'left
3610 IF X = 80 THEN GOTO 3850              'down
3620 IF X = 72 THEN GOTO 3900              'up
3630 'Whole page movements
3640 REM IF X=116 THEN GOTO 3970                'right 1 screen
3650 REM IF X=115 THEN GOTO 4030                'left 1 screen
3660 IF X = 81 THEN GOTO 4100              'down 1 screen
3670 IF X = 73 THEN GOTO 4160              'up 1 screen
3672 IF X = 71 THEN GOTO 4600              'home
3673 REM IF X=79 THEN GOTO 4650                 'end
3680 SOUND FREQ, DUR: GOTO 3490
3740 B$ = B$ + RIGHT$(X$, 1): GOTO 3490
3750 'Move 1 cell to the right
3760 GOSUB 4500: FG = 7: BG = 0: GOSUB 4800: GOSUB 4900: FG = 0: BG = 2:
GOSUB 4800
3770 GOTO 3490
3800 'Move 1 cell to the left
3810 GOSUB 4500: FG = 7: BG = 0: GOSUB 4800: GOSUB 4950: FG = 0: BG = 2:
GOSUB 4800
3820 GOTO 3490
3850 'Move 1 cell down to the bottom
3860 GOSUB 4500: FG = 7: BG = 0: GOSUB 4800: GOSUB 5000: COL = COLADJ +
ITAB(IFIELD): FG = 0: BG = 2: GOSUB 4800
3870 GOTO 3490
3900 'Move 1 cell up to the top
3910 GOSUB 4500: FG = 7: BG = 0: GOSUB 4800: GOSUB 5050: COL = COLADJ +
ITAB(IFIELD): FG = 0: BG = 2: GOSUB 4800
3920 GOTO 3490
3970 'Move 1 full page to the right
4030 'Move 1 full page to the left
4100 'Move 1 full page down
4110 GOSUB 4500: IPS = IPS + 30: IF IPS > IMAX THEN IPS = IMAX
4120 IF IPS = 0 THEN IPS = 1
4130 GOSUB 1200: GOTO 3490
4150 'Move 1 full page up
4160 GOSUB 4500: IPS = IPS - 30: IF IPS < 1 THEN IPS = 1
4170 GOSUB 1200: GOTO 3490
4500 REM store screen value to array
4510 IF LEN(B$) < 1 THEN RETURN
4520 IF IFIELD = 1 THEN ISURF$(IWANT) = LEFT$(B$ + "      ", NOCHR)
4530 IF IFIELD = 2 THEN ITREND$(IWANT) = B$
4540 IF IFIELD = 3 THEN IPLUNG$(IWANT) = B$
4550 IF IFIELD = 4 THEN IROCK$(IWANT) = LEFT$(B$ + "      ", NOCHR)
4555 IF IWANT > IMAX THEN IMAX = IWANT
4560 RETURN
4600 'Move to home position
4800 REM display present position
```

```
4805 IWANT = IPS + ROW - 6: IF COLADJ > 1 THEN IWANT = IWANT + 15
4810 NOCHR = ILEN(IFIELD): LOCATE ROW, COL: GOSUB 4850: COLOR FG,
BG: PRINT CELL$; : B$ = ""
4820 RETURN
4850 REM determined value in current field
4855 IF IFIELD = 1 THEN CELL$ = RIGHT$("      " + ISURF$(IWANT), NOCHR)
4860 IF IFIELD = 2 THEN CELL$ = RIGHT$("      " + ITREND$(IWANT), NOCHR)
4870 IF IFIELD = 3 THEN CELL$ = RIGHT$("      " + IPLUNG$(IWANT), NOCHR)
4880 IF IFIELD = 4 THEN CELL$ = RIGHT$("      " + IROCK$(IWANT), NOCHR)
4890 RETURN
4900 REM increment column position
4910 IFIELD = IFIELD + 1: IF IFIELD > 4 THEN IFIELD = 1: GOSUB 5000
4920 NOCHR = ILEN(IFIELD): COL = COLADJ + ITAB(IFIELD)
4930 RETURN
4950 REM decrement column position
4960 IFIELD = IFIELD - 1: IF IFIELD < 1 THEN IFIELD = 4: GOSUB 5050
4970 NOCHR = ILEN(IFIELD): COL = COLADJ + ITAB(IFIELD)
4980 RETURN
5000 REM increment row position
5010 ROW = ROW + 1: IF ROW > 20 THEN IF COLADJ = 0 THEN ROW = 6:
COLADJ = 40 ELSE ROW = 6: COLADJ = 0: SOUND FREQ, DUR
5020 RETURN
5050 REM decrement row position
5060 ROW = ROW - 1: IF ROW < 6 THEN IF COLADJ = 40 THEN ROW = 20:
COLADJ = 0 ELSE ROW = 6: SOUND FREQ, DUR
5070 RETURN
5100 REM repeat the last line
5110 IMAX = IMAX + 1: IF IMAX = 1 THEN IMAX = IMAX - 1: RETURN
5120 ISURF$(IMAX) = ISURF$(IMAX - 1): ITREND$(IMAX) = ITREND$(IMAX -
1): IPLUNG$(IMAX) = IPLUNG$(IMAX - 1): IROCK$(IMAX) = IROCK$(IMAX - 1)
5130 FOR I = 1 TO 4: FG = 7: BG = 0: GOSUB 4800: GOSUB 4900: FG = 0: BG = 2:
GOSUB 4800: NEXT I
5150 GOTO 3490
6000 REM initialize array values
6020 ILEN(1) = 2: ITAB(1) = 10
6030 IF IOPT = 1 THEN ILEN(2) = 5: ITAB(2) = 16 ELSE ILEN(2) = 3: ITAB(2) = 17
6040 IF IOPT = 1 THEN ILEN(3) = 4: ITAB(3) = 24 ELSE ILEN(3) = 2: ITAB(3) = 25
6050 ILEN(4) = 3: ITAB(4) = 31
6060 RETURN
6100 REM reduce all input data to dip and dip-direction
6110 DNAME$ = "D" + LEFT$(TNAME$ + "          ", 10)
6120 FOR I = 1 TO IMAX: IF IOPT = 1 THEN PLUNGE$ = IPLUNG$(I): STRIKE$ =
ITREND$(I): GOSUB 1000 ELSE DDIP = VAL(ITREND$(I)): DIP =
VAL(IPLUNG$(I))
6130 LBUF$ = DNAME$ + LEFT$(ISURF$(I), 2): LBUF$ = LBUF$ + RIGHT$("  " +
STR$(DIP), 2)
6140 LBUF$ = LBUF$ + RIGHT$("   " + STR$(DDIP), 3): LBUF$ = LBUF$ +
LEFT$(IROCK$(I), 3)
6150 PRINT #2, LBUF$: NEXT I
6160 RETURN
9000 REM==============( TERMINATE PROGRAM )=================
9010 CLOSE : PRINT : PRINT TAB(15); "END OF DATA INPUT"
9020 END
```

# PROGRAM BLOCK

# PROGRAM BLOCK

**Purpose:** To estimate the volume (size) distribution of a matrix block, formed by three joint sets and to define the average or typical shape factor for the block.

**Input:** The mean spacing of joints in each of three different joint sets.

**Output:**

1) The frequency and cumulative frequency distribution for block volume, together with summary statistics.
2) The frequency distribution of minimum and maximum side length.
3) A shape factor plot.

**Theory:** The program assumes that joint length is greater than joint spacing, therefore the matrix block can be defined by the appropriate spacings. It is also assumed that if more than three joint sets exist, the matrix block will be formed by joints from the three most frequently occurring sets. Joint spacing is assumed to follow the negative exponential distribution. Monte Carlo simulation is then used to generate different spacings within each of the three joint sets and thus to generate different block sizes. In this manner the distribution of block volume is obtained.

**Running:** The program block is entered through the underground series of programs:

1) First type UG. A disclaimer will then appear on the screen, Figure 1. A menu will then follow, Figure 2. Select item A, Rock Quality Analysis.

2) The Rock Quality menu will appear on the screen, Figure 3. Select item A, Block Program. A header sheet, Figure 4 will then follow.

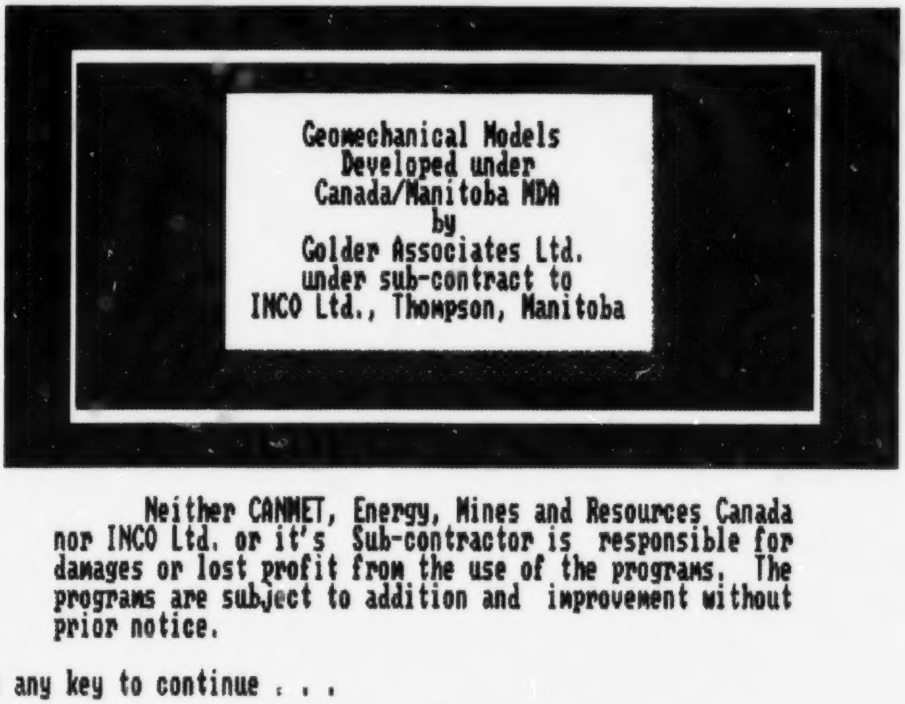
3) The first question asked is whether the same random number is required for each generation of joint spacing, Figure 5a). If you believe that the occurrence of each joint set is completely independent from the others, a different random number should be selected for each generation. If, however, there is a degree of interdependence then perhaps the same random number should be selected for the generation of the three block sides. (Note: This aspect requires further investigation.) Using both options should result in bounds to the expected block size.

4) Once the random number selection is made the screen will show a table, Figure 5b). Values for the mean spacing of each of the three joint sets can be entered via this table, Figure 5c). Note that the values for the mean dip and dip direction can also be entered. As

this part of the program is not operational at this time; blanks can therefore be entered, see Figure 5c.

5) Once all data is entered an overlay will appear over the table requesting information on the number of iterations required, Figure 6. Generally 100 is sufficient.

6) The various graphs and figures will then follow, Figures 7, 8 and 9.



Geomechanical Models
Developed under
Canada/Manitoba MDA
by
Golder Associates Ltd.
under sub-contract to
INCO Ltd., Thompson, Manitoba

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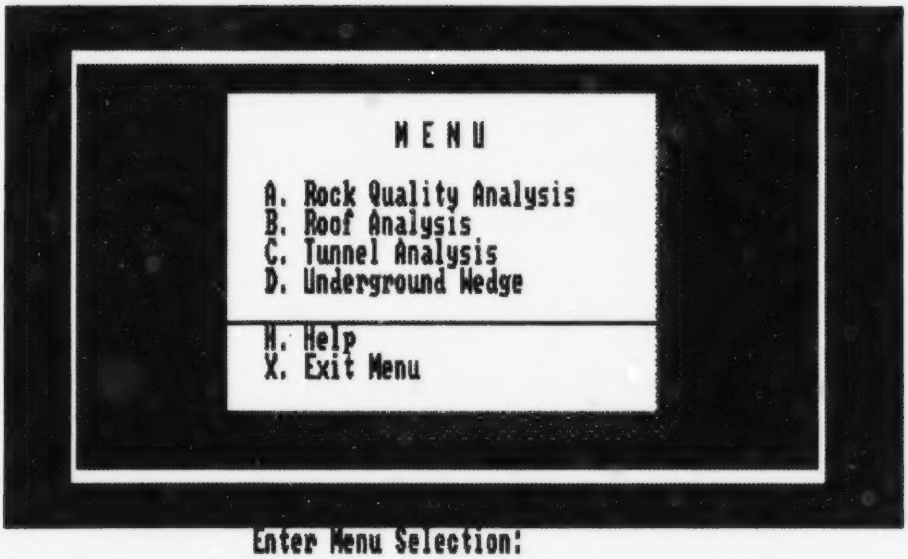
Press any key to continue . . .

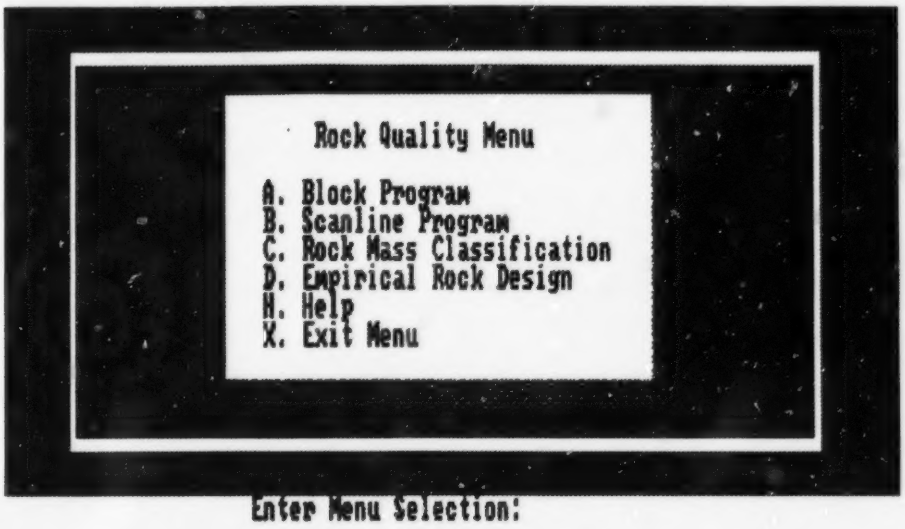
DISCLAIMER MESSAGE Figure 1


MENU
A. Rock Quality Analysis
B. Roof Analysis
C. Tunnel Analysis
D. Underground Wedge
H. Help
X. Exit Menu

Enter Menu Selection:

(Note programs A - Rock Quality, B - Roof Analysis, and C - Tunnel Analysis, have been developed as part of Task 1, VBM Design Guidelinese. Details of these programs given in Appendix entitled "Support Design" of the VBM guidelines).

MAIN MENU Figure 2



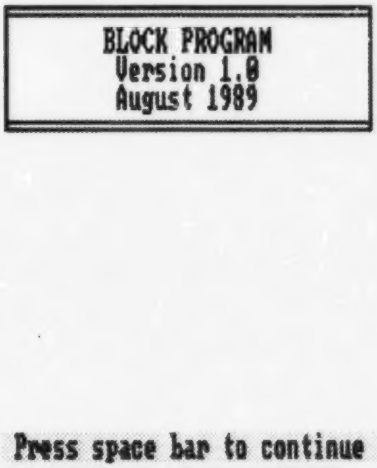
```
Rock Quality Menu

A. Block Program
B. Scanline Program
C. Rock Mass Classification
D. Empirical Rock Design
H. Help
X. Exit Menu

Enter Menu Selection:
```

(Note program C - Rock Mass Classification and D - Empirical Rock Design have been developed as part of Task 1, VBM Design Guidelines. Details of the program are given in the Appendix entitled "Support Design" of the VBM guidelines.)

**ROCK QUALITY MENU** Figure 3

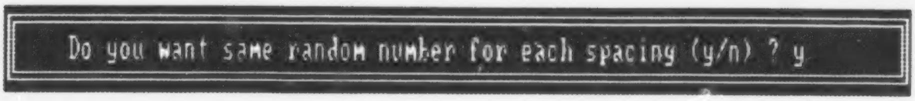


```
BLOCK PROGRAM
Version 1.0
August 1989

Press space bar to continue
```

**BLOCK HEADER SHEET** Figure 4

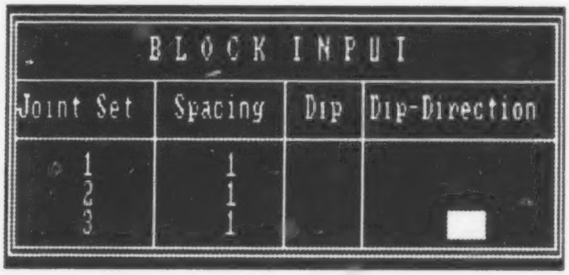
CANMET Series - BLOCK Program (version 1.0) - 09-01-1989 - 07:43:37



```
Do you want same random number for each spacing (y/n) ? y
```

RANDOM NUMBER GENERATOR Figure 5a

CANMET Series - BLOCK Program (version 1.0) - 09-01-1989 - 07:44:21

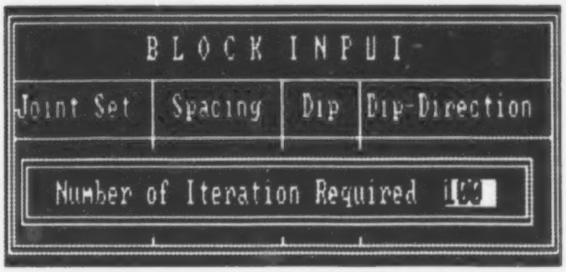


BLOCK INPUT

| Joint Set | Spacing | Dip | Dip-Direction |
|---|---|---|---|
| 1 | 1 | | |
| 2 | 1 | | |
| 3 | 1 | | |

ENTERING SPACING DATA Figure 5b

CANMET Series - BLOCK Program (version 1.0) - 09-01-1989 - 07:44:21



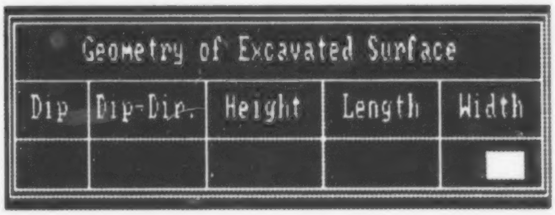
BLOCK INPUT

| Joint Set | Spacing | Dip | Dip-Direction |
|---|---|---|---|

Number of Iteration Required 100

NUMBER OF ITERATION REQUIRED Figure 6

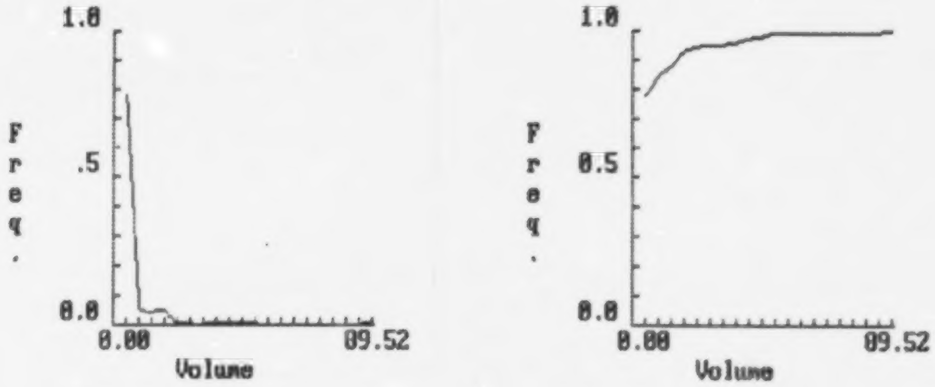
CANMET Series - BLOCK Program (version 1.0) - 09-01-1989 - 07:47:08



Geometry of Excavated Surface

| Dip | Dip-Dir. | Height | Length | Width |
|---|---|---|---|---|
| | | | | |

EXCAVATION DATA (NOT OPERATIONAL Figure 5c

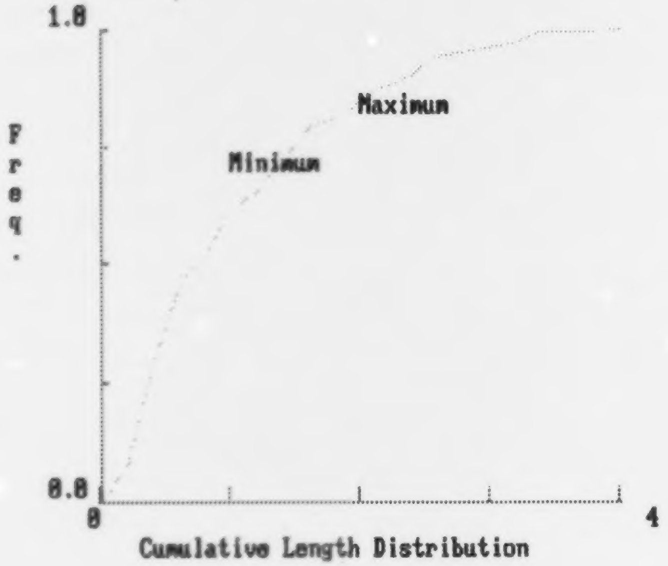


Frequency Distribution

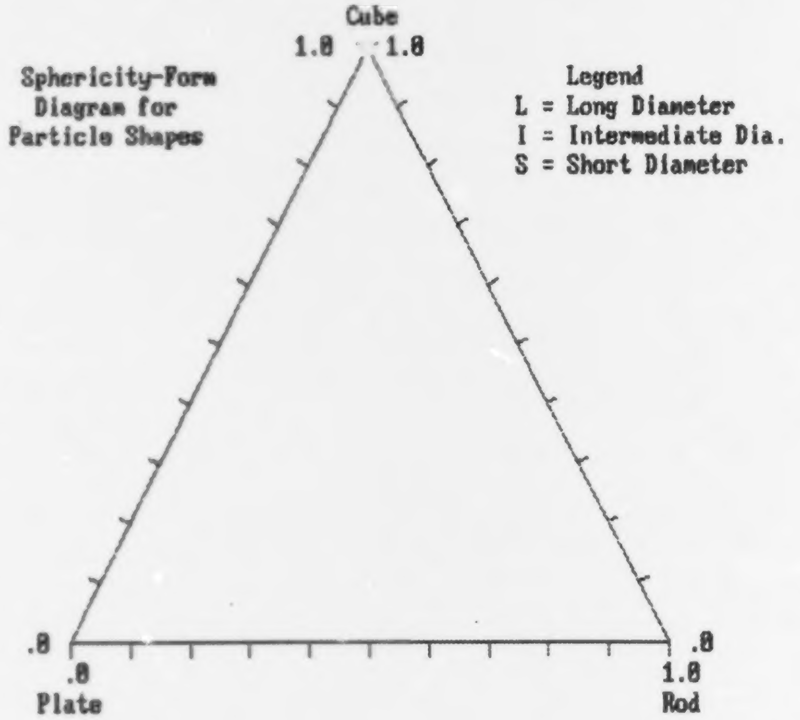
Cumulative Distribution

```
Mean Spacings  =    1.04,   1.04,   1.04   Mean Volume     =   5.14
Std. Deviation = %804290.28                Min/Max Volume =   0.00,  89.51
```

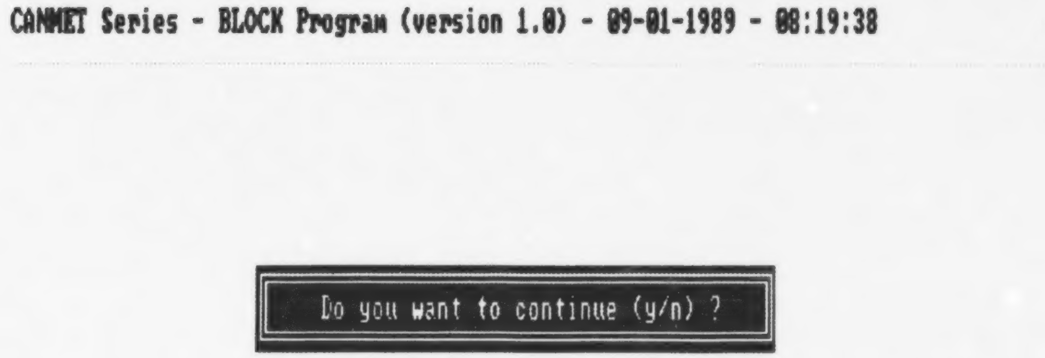
DISTRIBUTION OF BLOCK VALUES Figure 7



**DISTRIBUTION OF MAXIMUM AND MININUM SIDE LENGTHS OF BLOCK** Figure 8


Cube
1.0
1.0
Sphericity-Form
Diagram for
Particle Shapes
Legend
L = Long Diameter
I = Intermediate Dia.
S = Short Diameter
.0
.0
Plate
.0
1.0
Rod

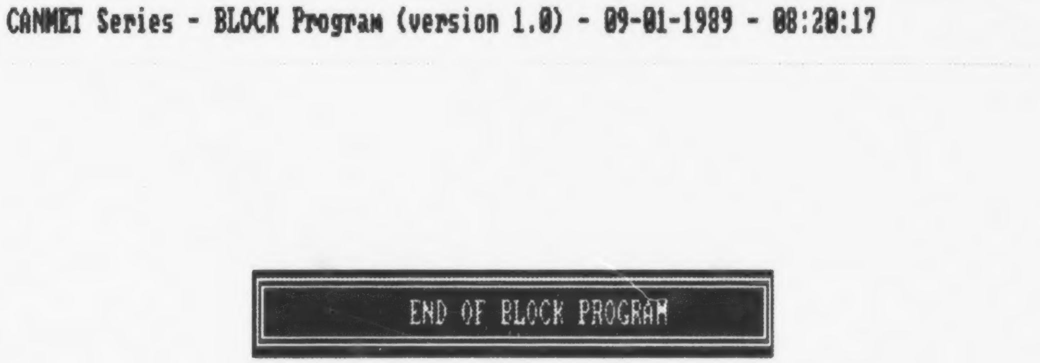
**SHAPE PLOT** Figure 9



```
CANMET Series - BLOCK Program (version 1.0) - 09-01-1989 - 08:19:38

Do you want to continue (y/n) ?
```

**CONTINUATION CONFIRMATION** Figure 10



```
CANMET Series - BLOCK Program (version 1.0) - 09-01-1989 - 08:20:17

END OF BLOCK PROGRAM
```

**PROGRAM TERMINATION** Figure 11

/am31605

# BLOCK PROGRAM LISTING

```
 DECLARE SUB PAUSE ()
 DECLARE SUB GetConfig ()
 DECLARE SUB SetConfig (mode AS INTEGER)
 DECLARE FUNCTION DegToRad! (Degrees AS INTEGER)
 DECLARE SUB SCALE ()
 DECLARE SUB CUM.DIST.LENGTH (SL#(), SS#(), nt%)
 DECLARE SUB MARK.SPOT (SL#(), SI#(), SS#(), NN%, RADIUS!)
 DECLARE SUB GNUMB (X!, Y!, X$, NOCHR%, B$, FG, BG)
 DECLARE SUB BOXED (X!, Y!, W!, D!, FG!, BG!)
 DECLARE SUB SAMEXP (X!, S#)
 DECLARE SUB GRAPH.FRAME ()
 DECLARE SUB FREQ.DIST.GRAPH (bin!, scal#, nt%)
 DECLARE SUB CUM.DIST.GRAPH (bin!, scal#, nt%)
 DECLARE SUB SPHERICITY.FRAME ()
 DECLARE SUB SAMLEN (SL#, SI#, SS#, S1#, S2#, S3#)
  DEFINT I-N: DEFDBL S-T, V
REM $DYNAMIC
  COMMON SHARED XARRAY(), EL, NPTS, MCNT, XMIN, XMAX, ymin, ymax,
scal, idec
  COMMON SHARED iscrn.type, ISCRN.COLOR, xscr1, yscr1, xscr2, yscr2
  COMMON SHARED bits.per.char, bits.per.line, x1, y1, x2, y2, X3, y3
  COMMON SHARED bin(), ILAST, CMODE
  COMMON SHARED VOL(), vmin, vmax
  DIM.XARRAY(4), bin(20), VOL(2000), SL(2000), SI(2000), SS(2000)
  DIM SMEAN(3), SPACING$(3), DIP$(3), DIPDIR$(3), HOR(4)
  DIM SPACING(3), DIP(3), DIPDIR(3), SSL(1), SSI(1), SSS(1)
' General purpose constants
 CONST PI = 3.14159
 CONST TRUE = -1, FALSE = 0
 CONST BACK = 0
 CONST TROW = 24, TCOL = 60

' Constants for best available screen mode
 CONST VGA = 9                    'Forced as EGA256
 CONST MCGA = 13
 CONST EGA256 = 9
 CONST EGA64 = 8
 CONST MONO = 10
 CONST HERC = 3
 CONST CGA = 2


' User-defined type to hold information about the mode
TYPE Config
  Scrn    AS INTEGER
  Colors  AS INTEGER
  Atribs  AS INTEGER
  XPix    AS INTEGER
  YPix    AS INTEGER
  TCOL     AS INTEGER
  TROW     AS INTEGER
END TYPE

DIM VC AS Config

' A palette of colors to paint with
DIM Pal(0 TO 300) AS LONG

' Error variables to check screen type
```

```
DIM InitRows AS INTEGER, BestMode AS INTEGER, Available AS STRING

 GOTO 10
'----------------------------------------------------------------------
'                    PRINT PAGE HEADER
'----------------------------------------------------------------------
2 SCREEN 0, 1: WIDTH 80: COLOR 7, 1: CLS
 PRINT "CANMET Series - BLOCK Program (version 1.0) - " + DATE$ + " - " +
TIME$
 COLOR 4, 1: PRINT STRING$(80, "_")
 COLOR 7, 1: RETURN
'----------------------------------------------------------------------
'                  PRINT PROGRAM HEADER
'----------------------------------------------------------------------
10 REM CANMET BLOCK (Version 1.00) - PROGRAM HEADER
30 REM Licensed Material - Program Property of GOLDER ASSOCIATES (WC)
LIMITED
40 KEY OFF: SCREEN 0, 1: COLOR 5, 0, 0: WIDTH 80: CLS :
60 COLOR 7, 1: LOCATE 10, 25, 0: PRINT CHR$(213) + STRING$(27, 205) +
CHR$(184)
70 LOCATE 11, 25, 0: PRINT CHR$(179) + "       BLOCK PROGRAM       " +
CHR$(179)
80 LOCATE 12, 25, 0: PRINT CHR$(179) + "       Version 1.0        " + CHR$(179)
90 LOCATE 13, 25, 0: PRINT CHR$(179) + "       August 1989        " + CHR$(179)
100 LOCATE 14, 25, 0: PRINT CHR$(212) + STRING$(27, 205) + CHR$(190)
120 COLOR 7, 4, 0: LOCATE 25, 25, 0: PRINT " Press space bar to continue ";
130 REM
140 CMD$ = INKEY$
150 IF CMD$ = "" THEN GOTO 140
170 IF CMD$ = " " THEN GOTO 190
180 GOTO 130
'----------------------------------------------------------------------
'                   DATA ARRAY DEFINITION
'----------------------------------------------------------------------
190 GOSUB 2
' Get best configuration and set initial graphics mode to it
 GetConfig
 VC.Scrn = BestMode
 SetConfig VC.Scrn
 bits.per.char = INT((VC.XPix + 1) / VC.TCOL)
 bits.per.line = INT((VC.YPix + 1) / VC.TROW)
 PLANES = VC.Atribs
 BITS.PER.PLANE = 1
 Scrn = VC.Scrn
 CMODE = VC.Colors
 XVU1 = 2 * bits.per.char
 ZVU1 = 2 * bits.per.line
 XVU2 = (VC.TCOL - 2) * bits.per.char
 ZVU2 = (VC.TROW - 2) * bits.per.line
 IEXIT = 0
 GOSUB 2
195 iscrn.type = VC.Scrn
  ISCRN.COLOR = 0
  xscr1 = 0
  yscr1 = 0
  xscr2 = VC.XPix
  yscr2 = VC.YPix
'----------------------------------------------------------------------
'                      MAIN MENU
```

```
'-----------------------------------------------------------------------
220 CALL BOXED(12, 5, 70, 3, 0, 7)
   LOCATE 13, 10: INPUT "Do you want same random number for each spacing (y/n) ";
RN$
   RN$ = LEFT$(RN$, 1)
   IF RN$ = "y" OR RN$ = "Y" THEN
         RN$ = "Y"
   ELSE
         RN$ = "N"
   END IF

   GOSUB 600: GOSUB 700: GOSUB 800: GOSUB 900
   GOSUB 1000
   GOSUB 1050
   GOSUB 1100
280 GOSUB 2: CALL BOXED(10, 20, 40, 3, 0, 7)
   LOCATE 11, 25: INPUT "Do you want to continue (y/n) "; an$
   an$ = LEFT$(an$, 1): IF an$ = "y" OR an$ = "Y" THEN GOTO 220
   GOTO FINISH

'-----------------------------------------------------------------------
'                     MAIN MENU OPTIONS
'-----------------------------------------------------------------------
600 REM
610 CLOSE : OPEN "DEBUGFL.BLK" FOR OUTPUT AS #3
   PRINT #3, "ITER    S1      S2      S3      Vol     S/L    (L-I)\(L-S)"
   CLS : GOSUB 2: COLOR 0, 7
   LOCATE 5, 21: PRINT CHR$(201) + STRING$(41, 205) + CHR$(187);
   LOCATE 6, 21: PRINT CHR$(186) + STRING$(10, 32) + "B L O C K  I N P U T" +
STRING$(11, 32) + CHR$(186);
   LOCATE 7, 21: PRINT CHR$(199) + STRING$(10, 196) + CHR$(194) +
STRING$(9, 196) + CHR$(194) + STRING$(5, 196) + CHR$(194) + STRING$(14,
196) + CHR$(182);
   LOCATE 8, 21: PRINT CHR$(199) + "Joint Set " + CHR$(179) + " Spacing " +
CHR$(179) + " Dip " + CHR$(179) + "Dip-Direction " + CHR$(186);
   LOCATE 9, 21: PRINT CHR$(199) + STRING$(10, 196) + CHR$(197) +
STRING$(9, 196) + CHR$(197) + STRING$(5, 196) + CHR$(197) + STRING$(14,
196) + CHR$(182);
   YT = 9: FOR n = 1 TO 3: LOCATE YT + n, 21: PRINT CHR$(186) + STRING$(10,
32) + CHR$(179) + STRING$(9, 32) + CHR$(179) + STRING$(5, 32) + CHR$(179) +
STRING$(14, 32) + CHR$(186);
   NEXT n: LOCATE 13, 21: PRINT CHR$(200) + STRING$(10, 205) + CHR$(207) +
STRING$(9, 205) + CHR$(207) + STRING$(5, 205) + CHR$(207) + STRING$(14,
205) + CHR$(188);
   COLOR 7, 0
REM ENTER BLOCK INPUT VALUES

630 XT = 26: YT = 9: FG = 0: BG = 7: COLOR FG, BG: FOR n = 1 TO 3
640 LOCATE YT + n, 26: PRINT n; : X = 36: Y = YT + n: NOCHR = 3
   CALL GNUMB(X, Y, X$, NOCHR, B$, FG, BG): SPACING$(n) = X$
641 X = 45: NOCHR = 2: CALL GNUMB(X, Y, X$, NOCHR, B$, FG, BG)
   IF ASC(B$) = 75 THEN GOTO 640
   DIP$(n) = X$: X = 55: NOCHR = 3: CALL GNUMB(X, Y, X$, NOCHR, B$, FG,
BG)
   IF ASC(B$) = 75 THEN GOTO 641
   DIPDIR$(n) = X$: NEXT n
650 A = 10: B = 22: W = 40: D = 3: FG = 0: BG = 7: CALL BOXED(A, B, W, D, FG,
BG)
   LOCATE 11, 25: PRINT "Number of Iteration Required ";
   X = 55: Y = 11: NOCHR = 4: CALL GNUMB(X, Y, X$, NOCHR, B$, FG, BG)
   IF ASC(B$) = 75 THEN GOTO 610 ELSE nt = VAL(X$)
660 GOSUB 2: A = 5: B = 20: W = 42: D = 7: CALL BOXED(A, B, W, D, FG, BG)
   LOCATE 6, 25: PRINT " Geometry of Excavated Surface";
   LOCATE 7, 20: PRINT CHR$(199) + STRING$(5, 196) + CHR$(194) + STRING$(8,
196) + CHR$(194) + STRING$(8, 196) + CHR$(194) + STRING$(8, 196) + CHR$(194)
+ STRING$(7, 196) + CHR$(182);
   LOCATE 8, 20: PRINT CHR$(186) + " Dip " + CHR$(179) + "Dip-Dir." +
CHR$(179) + " Height " + CHR$(179) + " Length " + CHR$(179) + " Width " +
CHR$(186);
   LOCATE 9, 20: PRINT CHR$(199) + STRING$(5, 196) + CHR$(197) + STRING$(8,
196) + CHR$(197) + STRING$(8, 196) + CHR$(197) + STRING$(8, 196) + CHR$(197)
+ STRING$(7, 196) + CHR$(182)
   LOCATE 10, 20: PRINT CHR$(186) + STRING$(5, 32) + CHR$(179) + STRING$(8,
32) + CHR$(179) + STRING$(8, 32) + CHR$(179) + STRING$(8, 32) + CHR$(179) +
STRING$(7, 32) + CHR$(186)
   LOCATE 11, 20: PRINT CHR$(200) + STRING$(5, 205) + CHR$(207) +
STRING$(8, 205) + CHR$(207) + STRING$(8, 205) + CHR$(207) + STRING$(8, 205)
+ CHR$(207) + STRING$(7, 205) + CHR$(188);
670 X = 23: Y = 10: NOCHR = 2: CALL GNUMB(X, Y, X$, NOCHR, B$, FG, BG)
   IF ASC(B$) = 75 THEN GOTO 650
675 D4$ = X$: X = 30: NOCHR = 3: CALL GNUMB(X, Y, X$, NOCHR, B$, FG, BG)
   IF ASC(B$) = 75 THEN GOTO 670
680 DD4$ = X$: X = 39: NOCHR = 3: CALL GNUMB(X, Y, X$, NOCHR, B$, FG,
BG)
   IF ASC(B$) = 75 THEN GOTO 675
685 HT$ = X$: X = 48: NOCHR = 3: CALL GNUMB(X, Y, X$, NOCHR, B$, FG, BG)
   IF ASC(B$) = 75 THEN GOTO 680
   LN$ = X$: X = 57: NOCHR = 3: CALL GNUMB(X, Y, X$, NOCHR, B$, FG, BG)
   IF ASC(B$) = 75 THEN GOTO 685
   WD$ = X$:
   RETURN
'-------------------------------------------------------------------------
'       CALCULATE EXPONENTIAL DISTRIBUTION DEFINED BY MEAN(X)
'-------------------------------------------------------------------------
700    SUM1 = 0!: SUM2 = 0!: SUM3 = 0!: SUMVOL = 0!: SUMVSQ = 0!
       RANDOMIZE TIMER
       S1 = VAL(SPACING$(1)): S2 = VAL(SPACING$(2)): S3 =
VAL(SPACING$(3))
       JCOUNT = nt / 20: J = 0: COLOR 4, 0: B$ = "."
       FOR I = 1 TO nt
       IF RN$ = "Y" THEN
       DUM = RND: TS1 = ABS(-S1 * LOG(DUM))
                  TS2 = ABS(-S2 * LOG(DUM))
                  TS3 = ABS(-S3 * LOG(DUM))
       ELSE
       DUM = RND: TS1 = ABS(-S1 * LOG(DUM))
       DUM = RND: TS2 = ABS(-S2 * LOG(DUM))
       DUM = RND: TS3 = ABS(-S3 * LOG(DUM))
       END IF

       CALL SAMLEN(SL(I), SI(I), SS(I), TS1, TS2, TS3)
       VOL(I) = TS1 * TS2 * TS3
       SDX = SL(I) - SS(I)
       IF SDX = 0! THEN SDX = .1
       S4 = SS(I) / SL(I): S5 = (SL(I) - SI(I)) / SDX
       PRINT #3, USING "####"; I;
       PRINT #3, USING "#######.##"; TS1; TS2; TS3; VOL(I); S4; S5
       SUM1 = SUM1 + TS1
       SUM2 = SUM2 + TS2
       SUM3 = SUM3 + TS3
```

```
        SUMVOL = SUMVOL + VOL(I)
        SUMVSQ = SUMVSQ + VOL(I) * VOL(I)
        IF J >= JCOUNT THEN B$ = B$ + ".": CLS : LOCATE 15, 10: PRINT B$ +
"One Moment Please"; : J = 0
        J = J + 1
        NEXT I
        RETURN
'--------------------------------------------------------------------
'               BUBBLE SORT OF ARRAY VOL
'--------------------------------------------------------------------
800     FLIPS = 1
        WHILE FLIPS
        FLIPS = 0
        FOR n = 1 TO nt - 1
        IF VOL(n) > VOL(n + 1) THEN SWAP VOL(n), VOL(n + 1): FLIPS = 1
        NEXT n
        WEND
        RETURN
'--------------------------------------------------------------------
'             DEFINE CLASSES AND COUNT FREQUENCY
'--------------------------------------------------------------------
900     vmin = 0!: vmax = VOL(nt)
        OPEN "DEBUG.OUT" FOR OUTPUT AS #2
        FOR n = 1 TO 20: bin(n) = 0: NEXT n
        RANGE = (vmax - vmin) / 20
        MAX.ENTRY = 0
        PRINT #2, "VMIN="; vmin; ", VMAX="; vmax; ", RANGE="; RANGE
        FOR n = 1 TO nt
        I = FIX((VOL(n) - vmin) / RANGE) + 1
        IF I > 20 THEN I = 20
        PRINT #2, "VOL("; n; ")="; VOL(n); ", IND = "; I
        bin(I) = bin(I) + 1
        NEXT n
        PRINT #2, "CONTENT OF BINS ="
        FOR n = 1 TO 20
        IF MAX.ENTRY < bin(n) THEN MAX.ENTRY = FIX(bin(n))
        PRINT #2, "BIN("; n; ")= "; bin(n)
        NEXT n
        PRINT #2, "MAXIMUM BIN COUNT = "; MAX.ENTRY: CLOSE #2
        RETURN
'--------------------------------------------------------------------
'               PLOT GRAPH SHOWING BLOCK VOLUME DISTRIBUTION
'                    AND SPHERICITY-FORM DIAGRAM
'--------------------------------------------------------------------
1000    SCREEN iscrn.type: ILAST = 0
        x1 = 1: y1 = bits.per.line
        x2 = xscr2: y2 = yscr2 - 2 * bits.per.line: CLS :
        IF iscrn.type <> 1 THEN VIEW SCREEN (x1, y1)-(x2, y2)
        IF CMODE > 0 THEN COLOR 1, 0
        LOCATE 1, 34: PRINT " BLOCK PROGRAM ";
        IF iscrn.type <> 1 THEN COLOR 4, 7
        CALL GRAPH.FRAME
        X = MAX.ENTRY / nt: scal = .1: ymin = 0!: idec = 1
        IF X < .5000916 THEN scal = .05: idec = 2
        IF X < .25 THEN scal = .025: idec = 3
        IF CMODE > 0 THEN COLOR 7, 0
        CALL FREQ.DIST.GRAPH(bin, scal, nt)
        scal = .1: ymin = 0!
        CALL CUM.DIST.GRAPH(bin, scal, nt)
        SMEAN(1) = SUM1 / nt: SMEAN(1) = FIX(SMEAN(1) * 100!) / 100!
        SMEAN(2) = SUM2 / nt: SMEAN(2) = FIX(SMEAN(2) * 100!) / 100!
        SMEAN(3) = SUM3 / nt: SMEAN(3) = FIX(SMEAN(3) * 100!) / 100!
        snt = nt
        vmean = SUMVOL / snt: vmean = FIX(vmean * 100!) / 100!
        SDEV = SQR(SUMVSQ / snt - (vmean * vmean)): SDEV = FIX(SDEV *
6554368!) / 100!
        vmin = FIX(vmin * 100!) / 100!: vmax = FIX(vmax * 100!) / 100!
        LOCATE 24, 1: PRINT "Mean Spacings  = ";
                     PRINT USING "####.##"; SMEAN(1); : PRINT ",";
                     PRINT USING "####.##"; SMEAN(2); : PRINT ",";
                     PRINT USING "####.##"; SMEAN(3);
        LOCATE 24, 45: PRINT "Mean Volume    = ";
                      PRINT USING "####.##"; vmean;
        LOCATE 25, 1: PRINT "Std. Deviation = ";
                      PRINT USING "####.##"; SDEV;
        LOCATE 25, 45: PRINT "Min/Max Volume = ";
                      PRINT USING "####.##"; vmin;
                      PRINT ","; : PRINT USING "####.##"; vmax;


MARK.TIME: X$ = INKEY$: IF X$ = "" THEN GOTO MARK.TIME
        RETURN
'-------------------------------------------------------------------------
'               PLOT GRAPH SHOWING BLOCK VOLUME DISTRIBUTION
'                     AND SPHERICITY-FORM DIAGRAM
'-------------------------------------------------------------------------
1050    SCREEN 1, 0: CLS : SCREEN iscrn.type: x1 = 1: y1 = bits.per.line
        x2 = xscr2: y2 = yscr2: CLS
        IF iscrn.type <> 1 THEN VIEW SCREEN (x1, y1)-(x2, y2)
        IF CMODE > 0 THEN COLOR 1, 0
        LOCATE 1, 34: PRINT " BLOCK PROGRAM "; : IF CMODE > 0 THEN
COLOR 7, 0
        CALL CUM.DIST.LENGTH(SL#(), SS#(), nt)
MARK.TIME1: X$ = INKEY$: IF X$ = "" THEN GOTO MARK.TIME1
        RETURN

'-------------------------------------------------------------------------
'                       SPHERICITY FORM DIAGRAM
'-------------------------------------------------------------------------
1100    SCREEN 1, 0: CLS : SCREEN iscrn.type: x1 = 1: y1 = bits.per.line
        x2 = xscr2: y2 = yscr2 - 2 * bits.per.line: CLS 0
        IF iscrn.type <> 1 THEN VIEW SCREEN (x1, y1)-(x2, y2)
        IF CMODE > 0 THEN COLOR 7, 0
        LOCATE 1, 36: PRINT " Cube ";
        LOCATE 2, 33: PRINT "1.0"; : LOCATE 2, 40: PRINT "1.0"
        LOCATE 3, 12: PRINT "Sphericity-Form";
        LOCATE 4, 13: PRINT "Diagram for";
        LOCATE 5, 11: PRINT "Particle Shapes";
        LOCATE 3, 54: PRINT "Legend";
        LOCATE 4, 50: PRINT "L = Long Diameter";
        LOCATE 5, 50: PRINT "I = Intermediate Dia.";
        LOCATE 6, 50: PRINT "S = Short Diameter";
        LOCATE 22, 12: PRINT ".0"; : LOCATE 22, 63: PRINT ".0";
        LOCATE 23, 15: PRINT ".0";
        LOCATE 23, 61: PRINT "1.0"
        LOCATE 24, 13: PRINT "Plate";
        LOCATE 24, 61: PRINT "Rod";
        RADIUS = 5: CALL SPHERICITY.FRAME
        CALL MARK.SPOT(SL(), SI(), SS(), nt, RADIUS)
        TS1 = SUM1 / nt
        TS2 = SUM2 / nt
        TS3 = SUM3 / nt
        CALL SAMLEN(SSSL, SSSI, SSSS, TS1, TS2, TS3)
        SSL(1) = SSSL: SSI(1) = SSSI: SSS(1) = SSSS
        IF iscrn.type <> 1 THEN COLOR 4, 0
        ILAST = 1
        CALL MARK.SPOT(SSL(), SSI(), SSS(), 1, RADIUS)
        ILAST = 0

        IF iscrn.type <> 1 THEN COLOR 7, 0
MARK.TIME.AGAIN: X$ = INKEY$: IF X$ = "" THEN GOTO MARK.TIME.AGAIN
        RETURN

'-------------------------------------------------------------------
'                     TERMINATE PROGRAM
'-------------------------------------------------------------------
FINISH:  CLOSE : GOSUB 2
 IF VC.Colors > 0 THEN COLOR 9
 A = 10: B = 20: W = 40: D = 3: IOPT = 1
 CALL BOXED(A, B, W, D, FG, BG): IF VC.Colors > 0 THEN COLOR 4
 LOCATE 11, 32: PRINT "END OF BLOCK PROGRAM"
 CALL PAUSE
 IF VC.Colors > 0 THEN COLOR 7
 END


' Error trap to make torus screen independent
VideoErr:
  SELECT CASE BestMode    ' Fall through until something works
    CASE VGA
        BestMode = MCGA
        Available = "12BD"
    CASE MCGA
        BestMode = EGA256
        Available = "12789"
    CASE EGA256
        BestMode = CGA
        Available = "12"
    CASE CGA
        BestMode = MONO
        Available = "A"
    CASE MONO
        BestMode = HERC
        Available = "3"
    CASE ELSE
        PRINT "Sorry. Graphics not available. Can't run Torus."
        END
  END SELECT
  RESUME

' Trap to detect 64K EGA
EGAErr:
  BestMode = EGA64
  Available = "12789"
  RESUME NEXT

' Trap to determine initial number of rows so they can be restored
RowErr:
  IF InitRows = 50 THEN
    InitRows = 43
    RESUME
  ELSE
    InitRows = 25
    RESUME NEXT
  END IF

REM $STATIC
SUB BOXED (X, Y, W, D, FG, BG) STATIC
'-------------------------------------------------------------------
'                  DRAW A DOUBLE-LINED BOX
'-------------------------------------------------------------------
COLOR FG, BG: LOCATE X, Y
PRINT CHR$(201) + STRING$(W - 2, 205) + CHR$(187);
  FOR L = X + 1 TO X + D - 2: LOCATE L, Y
    PRINT CHR$(186) + STRING$(W - 2, 32) + CHR$(186);
  NEXT L
LOCATE X + D - 1, Y: PRINT CHR$(200) + STRING$(W - 2, 205) + CHR$(188);
END SUB

SUB CUM.DIST.GRAPH (bin, scal, nt) STATIC

        x1 = 55 * bits.per.char: y1 = 16 * bits.per.line - 1
        XOFF = x1: YOFF = y1
        x2 = x1 + 20 * bits.per.char + 1: y2 = y1
        LINE (x1, y1)-(x2, y2)
        x2 = x1: y2 = y1 - 10 * bits.per.line
        LINE (x1, y1)-(x2, y2)
        LOCATE 18, 61: PRINT "Volume";
        LOCATE 10, 48: PRINT "F";
        LOCATE 11, 48: PRINT "r";
        LOCATE 12, 48: PRINT "e";
        LOCATE 13, 48: PRINT "q";
        LOCATE 14, 48: PRINT ".";
'Denote y-ticks
        FOR n = 1 TO 10
        x1 = XOFF: y1 = YOFF - n * bits.per.line
        x2 = x1 + .5 * bits.per.char: y2 = y1
        LINE (x1, y1)-(x2, y2)
        NEXT n
        ymin$ = "0.0": ymin$ = RIGHT$("      " + ymin$, 6)
        LOCATE 16, 49: PRINT ymin$;
        ymid$ = "0.5": ymid$ = RIGHT$("      " + ymid$, 6)
        LOCATE 11, 49: PRINT ymid$
        ymax$ = "1.0": ymax$ = RIGHT$("      " + ymax$, 6)
        LOCATE 6, 49: PRINT ymax$;
'Denote x-ticks
        FOR n = 1 TO 20
        x1 = XOFF + n * bits.per.char: y1 = YOFF
        x2 = x1: y2 = YOFF - .25 * bits.per.line
        LINE (x1, y1)-(x2, y2)
        NEXT n
        LOCATE 17, 52: PRINT USING "####.##"; vmin;
        LOCATE 17, 72: PRINT USING "####.##"; vmax;
'Plot distribution curve
        cum.freq = bin(1)
        FOR n = 2 TO 20
        x1 = XOFF + (n - 1) * bits.per.char
        y1 = YOFF - ((cum.freq / nt) / scal) * bits.per.line
        x2 = XOFF + n * bits.per.char
        cum.freq = cum.freq + bin(n)
        y2 = YOFF - ((cum.freq / nt) / scal) * bits.per.line
        LINE (x1, y1)-(x2, y2)
        NEXT n
        LOCATE 20, 55: PRINT "Cumulative Distribution";
END SUB

SUB CUM.DIST.LENGTH (SL#(), SS#(), nt) STATIC
        FOR I = 1 TO 20: bin(I) = 0!: NEXT I
        LINE (x1, y1)-(x2, y1), 4
        LINE (x2, y1)-(x2, y2), 4
        LINE (x2, y2)-(x1, y2), 4
        LINE (x1, y2)-(x1, y1), 4
        smx = 1E-20
        smn = 0!
        FOR n = 1 TO nt
        IF SL(n) > smx THEN smx = SL(n)
        NEXT n
        dist = smx / 20
        FOR n = 1 TO nt
        ii = SL(n) / dist + 1
        IF ii > 20 THEN ii = 20
        bin(ii) = bin(ii) + 1
        NEXT n
' Calculate Y Scale
        XARRAY(1) = smn
        XARRAY(2) = smx
        xscal = dist
        XMN = smn
        XMX = smx + 20! * xscal
' Calculate y Scale
        yscal = .25
        ymn = 0!
        ymx = 1!
' Plot x-axis and label.
        x1 = 21 * bits.per.char: y1 = 22 * bits.per.line - 1
        XOFF = x1: YOFF = y1
        x1 = XOFF: y1 = YOFF: x2 = x1 + 40 * bits.per.char: y2 = y1
        LINE (x1, y1)-(x2, y2), 14
        FOR n = 0 TO 40
        ii = n MOD 10
        IF ii > 0 THEN GOTO NXT.N.1
        x1 = XOFF + n * bits.per.char - 1: y2 = y1
        LINE (x1, y1)-(x2, y2), 14
        x2 = x1: y2 = y1 - .5 * bits.per.line
        LINE (x1, y1)-(x2, y2), 14
NXT.N.1:
        NEXT n
        LOCATE 10, 15: PRINT "F";
        LOCATE 11, 15: PRINT "r";
        LOCATE 12, 15: PRINT "e";
        LOCATE 13, 15: PRINT "q";
        LOCATE 14, 15: PRINT ".";
        LOCATE 15, 15: PRINT " ";
' Denote y-ticks
        x1 = XOFF: y1 = YOFF: x2 = x1: y2 = YOFF - 16 * bits.per.line
        LINE (x1, y1)-(x2, y2), 14
        FOR n = 0 TO 16
        ii = n MOD 4: IF ii > 0 THEN GOTO NXT.N
        x1 = XOFF: y1 = YOFF - n * bits.per.line
        x2 = x1 + .5 * bits.per.char: y2 = y1
        LINE (x1, y1)-(x2, y2), 14
NXT.N:
        NEXT n
        ymin$ = "0.0": ymin$ = RIGHT$("      " + ymin$, 6)
        LOCATE 22, 15: PRINT ymin$;
        ymax$ = "1.0": ymax$ = RIGHT$("      " + ymax$, 6)
        LOCATE 6, 15: PRINT ymax$;
'Plot cumulated maximum length distribution curve
        cumx = 0
        FOR n = 1 TO 20
        x1 = XOFF + (n - 1) * 2 * bits.per.char
        y1 = YOFF - (cumx / nt) / yscal * 4 * bits.per.line
        x2 = XOFF + n * 2 * bits.per.char
        cumx = cumx + bin(n)
        y2 = YOFF - cumx / nt / yscal * 4 * bits.per.line
        LINE (x1, y1)-(x2, y2), 4
        IF n = 10 THEN
            X = x2 / bits.per.char + 1
            Y = y2 / bits.per.line + 1
            LOCATE Y, X: PRINT "Maximum";
        END IF
        NEXT n

'Plot cumulated minimum length distribution curve
        FOR n = 1 TO 20: bin(n) = 0!: NEXT n
        FOR n = 1 TO nt
        ii = SS(n) / dist + 1
        IF ii > 20 THEN ii = 20
        bin(ii) = bin(ii) + 1
        NEXT n
        cumx = 0
        FOR n = 1 TO 20
        x1 = XOFF + (n - 1) * 2 * bits.per.char
        y1 = YOFF - cumx / nt / yscal * 4 * bits.per.line
        x2 = XOFF + n * 2 * bits.per.char
        cumx = cumx + bin(n)
        y2 = YOFF - cumx / nt / yscal * 4 * bits.per.line
        LINE (x1, y1)-(x2, y2), 2
        IF n = 5 THEN
            X = x2 / bits.per.char + 1
            Y = y2 / bits.per.line - 1
            LOCATE Y, X: PRINT "Minimum";
        END IF
        NEXT n
        LOCATE 23, 21: PRINT "0": LOCATE 23, 60: PRINT USING "#####"; smx
        LOCATE 24, 25: PRINT "Cumulative Length Distribution";
END SUB

SUB FREQ.DIST.GRAPH (bin, scal, nt) STATIC

        x1 = 15 * bits.per.char: y1 = 16 * bits.per.line - 1
        XOFF = x1: YOFF = y1
        x2 = x1 + 20 * bits.per.char + 1: y2 = y1
        LINE (x1, y1)-(x2, y2)
        x2 = x1: y2 = y1 - 10 * bits.per.line
        LINE (x1, y1)-(x2, y2)
        LOCATE 18, 21: PRINT "Volume";
        LOCATE 10, 8: PRINT "F";
        LOCATE 11, 8: PRINT "r";
        LOCATE 12, 8: PRINT "e";
```

```
          LOCATE 13, 8: PRINT "q";
          LOCATE 14, 8: PRINT ".";
'Denote y-ticks
          FOR n = 1 TO 10
          x1 = XOFF: y1 = YOFF - n * bits.per.line
          x2 = x1 + .5 * bits.per.char: y2 = y1
          LINE (x1, y1)-(x2, y2)
          NEXT n
          ymin$ = "0.0": ymin$ = RIGHT$("      " + ymin$, 6)
          LOCATE 16, 9: PRINT ymin$;
          ymid$ = STR$(ymin + 5! * scal): X = INSTR(ymid$, ".")
          IF X > 0 THEN ymid$ = LEFT$(ymid$, X + idec)
          ymid$ = RIGHT$("      " + ymid$, 6)
          LOCATE 11, 9: PRINT ymid$
          ymax$ = STR$(ymin + 10! * scal): X = INSTR(ymax$, ".")
          IF X > 0 THEN ymax$ = LEFT$(ymax$, X + idec)
          ymax$ = RIGHT$("      " + ymax$, 6)
          LOCATE 6, 9: PRINT ymax$;
'Denote x-ticks
          FOR n = 1 TO 20
          x1 = XOFF + n * bits.per.char: y1 = YOFF
          x2 = x1: y2 = YOFF - .25 * bits.per.line
          LINE (x1, y1)-(x2, y2)
          NEXT n
          LOCATE 17, 12: PRINT USING "####.##"; vmin;
          LOCATE 17, 32: PRINT USING "####.##"; vmax;
'Plot distribution curve
          FOR n = 2 TO 20
          x1 = XOFF + (n - 1) * bits.per.char
          y1 = YOFF - ((bin(n - 1) / nt) / scal) * bits.per.line
          x2 = XOFF + n * bits.per.char
          y2 = YOFF - ((bin(n) / nt) / scal) * bits.per.line
          LINE (x1, y1)-(x2, y2)
          NEXT n
          LOCATE 20, 15: PRINT "Frequency Distribution";
END SUB

' =========================== GetConfig
===============================
'  Get the starting number of lines and the video adapter.
'
==================================================================
======
'

SUB GetConfig STATIC
SHARED InitRows AS INTEGER, BestMode AS INTEGER, Available AS STRING

  ' Assume 50 line display and fall through error
  ' until we get the actual number
  InitRows = 50
  ON ERROR GOTO RowErr
  LOCATE InitRows, 1

  ' Assume best possible screen mode
  BestMode = VGA
  Available = "12789BCD"

  ON ERROR GOTO VideoErr
  ' Fall through error trap until a mode works
  SCREEN BestMode
  ' If EGA, then check pages to see whether more than 64K
  ON ERROR GOTO EGAErr
  IF BestMode = EGA256 THEN SCREEN 8, , 1

  ON ERROR GOTO 0

  ' Reset text mode
  SCREEN 0, , 0
  WIDTH 80, 25

END SUB

SUB GNUMB (X, Y, X$, NOCHR, B$, FG, BG) STATIC
REM ==================< NUMBER KEYS - SINGLE KEY ENTRY
>========================
C$ = STRING$(NOCHR, 32): LOCATE Y, X: COLOR BG, FG: PRINT C$: X$ = "":
B$ = " ": IFRST = 0
GNUMB1: A$ = INKEY$: IF A$ = "" THEN GOTO GNUMB1
IF LEN(A$) > 1 THEN B$ = RIGHT$(A$, 1): GOTO GNUMB9
IF ASC(A$) = 8 THEN GOTO GNUMB3                  ' Backspace key
IF ASC(A$) = 13 THEN GOTO GNUMB9                 ' Return key
IF ASC(A$) = 27 THEN GOTO GNUMB8                 ' Escape key
IF ASC(A$) = 46 THEN GOTO GNUMB2                 ' Decimal key
IF ASC(A$) = 45 AND IFRST = 0 THEN GOTO GNUMB2      ' Minus key
IF ASC(A$) = 43 AND IFRST = 0 THEN GOTO GNUMB2      ' Plus key
IF ASC(A$) > 47 AND ASC(A$) < 58 THEN GOTO GNUMB2  ' Number key
IF ASC(A$) = 47 THEN IFRST = -1: GOTO GNUMB2        ' Slash key
GOTO GNUMB1
GNUMB2: X$ = X$ + A$: LOCATE Y, X: PRINT X$; : IFRST = IFRST + 1
IF LEN(X$) = NOCHR THEN GOTO GNUMB9 ELSE GOTO GNUMB1
GNUMB3: NL = LEN(X$): IF NL > 1 THEN X$ = LEFT$(X$, NL - 1) ELSE X$ = ""
LOCATE Y, X: PRINT C$; : LOCATE Y, X: PRINT X$; : GOTO GNUMB1
GNUMB8: X$ = A$: GOTO GEXIT
GNUMB9: LOCATE Y, X: COLOR FG, BG: PRINT STRING$(NOCHR, 32):
LOCATE Y, X
PRINT RIGHT$(STRING$(NOCHR, 32) + X$, NOCHR);
GEXIT: END SUB

SUB GRAPH.FRAME STATIC

REM ============================< GRAPH.FRAME
>=============================

          x1 = xscr1 + 1: y1 = yscr1 + bits.per.line
          x2 = x1: y2 = yscr2 - 2 * bits.per.line - 1
          LINE (x1, y1)-(x2, y2)
          y1 = y2: x2 = xscr2
          LINE (x1, y2)-(x2, y2)
          x1 = x2: y1 = y2: y2 = yscr1 + bits.per.line
          LINE (x1, y1)-(x2, y2)
          x1 = x2: y1 = y2: x2 = xscr1 + 1
          LINE (x1, y1)-(x2, y2)
          x1 = FIX((xscr2 - xscr1) / 2): y1 = y2: x2 = x1: y2 = yscr2 - 2 * bits.per.line - 1
          LINE (x1, y1)-(x2, y2)

END SUB

SUB MARK.SPOT (SL(), SI(), SS(), nt, RADIUS) STATIC
     FOR I = 1 TO nt
```

```
        DX21 = x2 - x1: DY13 = y1 - y3
        SL.RATIO = SS(I) / SL(I)
        SDX = SL(I) - SS(I)
        IF SDX = 0! THEN SDX = .1
        DXL = x1 + ((SL(I) - SI(I)) / SDX) * DX21
        XC = DXL + (X3 - DXL) * SL.RATIO
        YC = y1 - SL.RATIO * DY13
        IXC = INT(XC): IYC = INT(YC)
        CIRCLE (IXC, IYC), RADIUS
        IF iscrn.type <> 1 OR ILAST = 1 THEN PAINT (IXC, IYC)
        NEXT I
END SUB

DEFINT A-H, O-Z
SUB PAUSE STATIC

CONT.PAUSE:

    X$ = INKEY$: IF X$ = "" THEN GOTO CONT.PAUSE

END SUB

DEFSNG A-H, O-R, U, W-Z
DEFDBL S-T, V
'***********************************************************************
*****
'               UTILITY ROUTINES
'***********************************************************************
*****
SUB SAMEXP (X, S) STATIC
'------------------------------------------------------------------------
'               SAMEXP ROUTINE
'------------------------------------------------------------------------

        DUM = RND
        S = ABS(-X * LOG(DUM))
END SUB

SUB SAMLEN (SL, SI, SS, S1, S2, S3) STATIC
SL = S1: IF SL < S2 THEN SL = S2
        IF SL < S3 THEN SL = S3
SS = S1: IF SS > S2 THEN SS = S2
        IF SS > S3 THEN SS = S3
SI = S1
IF S1 >= S2 AND S1 <= S3 THEN SI = S1
IF S1 >= S3 AND S1 <= S2 THEN SI = S1
IF S2 >= S1 AND S2 <= S3 THEN SI = S2
IF S2 >= S3 AND S2 <= S1 THEN SI = S2
IF S3 >= S1 AND S3 <= S2 THEN SI = S3
IF S3 >= S2 AND S2 <= S1 THEN SI = S3
END SUB

SUB SCALE STATIC
'   *******************************************************************
'   *                                 *
'   *       GOLDER ASSOCIATES LIBRARY ROUTINE         *
'   *                                 *
'   *   THIS ROUTINE IS THE PROPERTY OF GOLDER ASSOCIATES     *
'   *                                 *
'   *            SUBROUTINE SCALE            *
'   *                                              *
'   ******************************************************************
'
' THIS SUBROUTINE DOES THE SAME THING AS THE UNIVAC "SCALE"
ROUTINE,
'   (MORE OR LESS).
'
' ARRAY XARRAY CONTAINS THE VALUES TO BE SCALED.
' EVERY M"TH VALUE IS TO BE SCALED.
' THE AXIS LENGTH IS EL.
' THE VALUE AT THE START OF THE AXIS IS RETURNED IN XARRAY(N+1)
' THE VALUE OF THE SCALE FACTOR IS RETURNED IN XARRAY(N+2)
    FSCALE = 1
    DEL = EL
    XMAX = -1E+20
    XMIN = 1E+20
    n = NPTS: M = MCNT
    FOR I = 1 TO n STEP M
    IF XMAX < XARRAY(I) THEN XMAX = XARRAY(I)
    IF XMIN > XARRAY(I) THEN XMIN = XARRAY(I)
    NEXT I
    dist = (XMAX - XMIN) / EL
' TAKE THE LOG(10) OF THE DISTANCE.
5    A = LOG(dist) / 2.3025851# + 100!
' SPLIT OFF THE ARGUMENT OF THE LOG.
    B = A - FIX(A)
' ROUND THE ARGUMENT UP TO CORRESPOND TO AN EVEN SORT OF
NUMBER.
    C = 10!
    IF B < .7 THEN C = 5!
    IF FSCALE = 1 AND B < .602 THEN C = 4!
    IF B < .301 THEN C = 2!
    GSCALE = C * 10! ^ (A - B - 100!)
    XMIN = GSCALE * FIX(XMIN / GSCALE)
    IF XMIN < 0! THEN XMIN = XMIN - GSCALE
    IF (XMAX - XMIN) / GSCALE <= EL THEN GOTO 4
    DEL = DEL - 1!
    IF DEL <= 0! THEN PRINT "SCALE - LENGTH OF AXIS LESS THAN 1": STOP
    dist = (XMAX - XMIN) / DEL
    GOTO 5
4    XARRAY(n + 1) = XMIN
    XARRAY(n + 2) = GSCALE
    FOR I = 1 TO n STEP M
    XARRAY(I) = (XARRAY(I) - XMIN) / GSCALE
    NEXT I
END SUB

DEFINT A-H, O-Z
'============================= SetConfig
=================================
'  Sets the correct values for each field of the VC variable. They
'  vary depending on Mode and on the current configuration.
'
=================================================================
======
'

SUB SetConfig (mode AS INTEGER) STATIC
SHARED VC AS Config, BestMode AS INTEGER

  SELECT CASE mode
```

```
CASE 1  ' Four-color graphics for CGA, EGA, VGA, and MCGA
    IF BestMode = CGA OR BestMode = MCGA THEN
      VC.Colors = 0
    ELSE
      VC.Colors = 16
    END IF
    VC.Atribs = 4
    VC.XPix = 319
    VC.YPix = 199
    VC.TCOL = 40
    VC.TROW = 25
CASE 2  ' Two-color medium-res graphics for CGA, EGA, VGA, and MCGA
    IF BestMode = CGA OR BestMode = MCGA THEN
      VC.Colors = 0
    ELSE
      VC.Colors = 16
    END IF
    VC.Atribs = 2
    VC.XPix = 639
    VC.YPix = 199
    VC.TCOL = 80
    VC.TROW = 25
CASE 3  ' Two-color high-res graphics for Hercules
    VC.Colors = 0
    VC.Atribs = 2
    VC.XPix = 720
    VC.YPix = 348
    VC.TCOL = 80
    VC.TROW = 25
CASE 7  ' 16-color medium-res graphics for EGA and VGA
    VC.Colors = 16
    VC.Atribs = 16
    VC.XPix = 319
    VC.YPix = 199
    VC.TCOL = 40
    VC.TROW = 25
CASE 8  ' 16-color high-res graphics for EGA and VGA
    VC.Colors = 16
    VC.Atribs = 16
    VC.XPix = 639
    VC.YPix = 199
    VC.TCOL = 80
    VC.TROW = 25
CASE 9  ' 16- or 4-color very high-res graphics for EGA and VGA
    VC.Colors = 64
    IF BestMode = EGA64 THEN VC.Atribs = 4 ELSE VC.Atribs = 16
    VC.XPix = 639
    VC.YPix = 349
    VC.TCOL = 80
    VC.TROW = 25
CASE 10 ' Two-color high-res graphics for EGA or VGA monochrome
    VC.Colors = 0
    VC.Atribs = 2
    VC.XPix = 319
    VC.YPix = 199
    VC.TCOL = 80
    VC.TROW = 25
CASE 11 ' Two-color very high-res graphics for VGA and MCGA
    ' Note that for VGA screens 11, 12, and 13, more colors are
    ' available, depending on how the colors are mixed.
```

```
        VC.Colors = 216
        VC.Atribs = 2
        VC.XPix = 639
        VC.YPix = 479
        VC.TCOL = 80
        VC.TROW = 30
    CASE 12  ' 16-color very high-res graphics for VGA
        VC.Colors = 216
        VC.Atribs = 16
        VC.XPix = 639
        VC.YPix = 479
        VC.TCOL = 80
        VC.TROW = 30
    CASE 13  ' 256-color medium-res graphics for VGA and MCGA
        VC.Colors = 216
        VC.Atribs = 256
        VC.XPix = 639
        VC.YPix = 479
        VC.TCOL = 40
        VC.TROW = 25
    CASE ELSE
        VC.Colors = 16
        VC.Atribs = 16
        VC.XPix = 0
        VC.YPix = 0
        VC.TCOL = 80
        VC.TROW = 25
        VC.Scrn = 0
        EXIT SUB
  END SELECT
  VC.Scrn = mode

END SUB

DEFSNG A-H, O-R, U, W-Z
DEFDBL S-T, V
SUB SPHERICITY.FRAME STATIC

REM =======================< SPHERICITY.FRAME
>==============================

          x1 = 116: y1 = yscr2 - bits.per.line * 3.5
          x2 = 484: y2 = y1
          X3 = x1 + (x2 - x1) / 2: y3 = yscr1 + 1.5 * bits.per.line
          LINE (x1, y1)-(x2, y2)
          LINE (x2, y2)-(X3, y3)
          LINE (X3, y3)-(x1, y1)
          xinc = (x2 - x1) / 10
'tick marks along bottom
          FOR I = 0 TO 10
          X = x1 + I * xinc
          LINE (X, y1)-(X, y1 + .5 * bits.per.line)
          NEXT I
'tick along left side
           xinc = (X3 - x1) / 10: yinc = (y3 - y1) / 10
           FOR I = 1 TO 10
           X = x1 + I * xinc
           Y = y1 + I * yinc
           LINE (X, Y)-(X - .75 * bits.per.char, Y - .3 * bits.per.char)
           NEXT I
```

```
'tick along right side
          xinc = (X3 - x2) / 10: yinc = (y3 - y2) / 10
          FOR I = 1 TO 10
          X = x2 + I * xinc
          Y = y2 + I * yinc
          LINE (X, Y)-(X + .75 * bits.per.char, Y - .3 * bits.per.char)
          NEXT I
END SUB
```